Dressing Weakly Magnetic Iron Ores by Dry Magnetic Separation Process

By

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ABSTRACT

Iron ore samples were obtained from El Bajrawiya iron ore deposits in order to beneficiate these ores. Chemical analyses and microscopic examination have shown that these ores mainly contain valuable mixtures of weakly magnetic iron oxides; (goethite, limonite and hematite), and non magnetic gangue materials; (silica), and clay minerals as major impurities. These materials were prepared to be beneficiated by dry high intensity magnetic separation process, by crushing and grinding to liberate the iron oxide grains from the cement of the ore matrix. Crushing and grinding processes produced significant amounts of fine materials (-53μm), which could not be treated by the available machine within the Central Petroleum Laboratories (dry high intensity magnetic separator). Hence, all ground samples materials were washed to remove the fines by wet screening on the 53 micron sieve. The portions of each sample coarser than 53 μm were dried and screened on a nest of sieves to yield four size fractions each having closer ranges for the magnetic separation in order to avoid the gravity effect on the process.

The Frantz Magnetic Barrier Laboratory Separator Model (LB-1) was used to conduct the separation tests on each size fractions of the washed various ores. Recovery of iron values and the separation efficiency were assessed by determining the percentages of Fe in the concentrates and tailings by Atomic Absorption Spectrometry (AAS) techniques..

The separation process was applied on four mixtures of iron oxides as feed for the process. They were having average grades 32.94, 41.43, 17.00 and 48.00% Fe. The process of treatment gave concentrates having average grades from the feed 39.79, 49.82, 47.67and 52.76 % Fe and average iron recoveries were 86.83, 87.63, 76.32 and 83.51 %, respectively. The flow rates of the feed into the separator were: 19.63, 20.32, 25.66 and 12.37 g / minute, respectively. These results were achieved when the Frantiz separator was operated at 0.8 ampere. However, when the ampere of the separator was increased to 1.0 ampere, the recoveries were increased on the account of the grades which were decreased remarkably.

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موجز البحث (الخلاصة)

تم الحصول علي عينات خام الحديد من ترسيبات خام الحديد بمنطقة البجراوية من اجل رفع درجة هذه الخامات. التحاليل الكيميائية والاختبارات المجهرية أوضحت أن هذه الخامات فى الاساس تحتوى على خلائط قيمة من اكاسيد الحديد المغنطيسية الضعيفة (الجيوسيت والليمونيت والهيماتيت) وعلى مواد غير مغنطيسية (سليكا) ومواد طينية كشوائب رئيسية. تم تجهيز هذه المواد لكى يتم تحسينها بواسطة عملية الفصل المغنطيسى الجاف ذو الشدة العالية، عن طريق التكسير والطحن لتحرير حبيبات اكاسيد الحديد من معادن الصخور الملتصقة بها. عمليتى التكسير والطحن أنتجت كميات ذات معنى من المواد الناعمة جدا(أقل من53 ميكرون) و التى لايمكن معالجتها بواسطة الاله الموجودة فى مختبرات النفط المركزية (جهاز الفصل المغنطيسى الجاف ذو الشدة العالية). لهذا السبب جميع العينات المطحونة تم غسلها، لازالة الحبيبات الناعمة جدا بواسطة الغربلة الرطبة على منخل فتحته53 ميكرون. الاجزاء الخشنه لكل عينة ، اكبر من53 ميكرون تم تجفيفها وغربلتها على مجموعة مناخل لانتاج اربعة كسور حجمية كل منها يمتلك حجوم متقاربة لعملية الفصل المغنطيسى من اجل تجنب اثر الجاذبية على عملية الفصل .

استخدم جهاز فصل مغنطيسى معملى يدعى فرانتز من طراز (LB-1) لإجراء تجارب الفصل المغنطيسى على كل الكسور الحجمية من الخامات المختلفة المغسولة. تم تقييم كفاءة الفصل وقيم العائدات الحديدية بواسطة تحديد النسب المؤية للحديد فى المركزات والنفايات بواسطة تقنية الامتصاص الذرى.

عملية الفصل المغنطيسى الجاف تم تطبيقها على اربعة خلائط من اكاسيد الحديد كغذاء للعملية. متوسط تحاليلها 48.00، 17.00, 41.43 و 32.94 فى المائة حديد . عملية المعالجة اعطت مركزات من الغذاء متوسط تحاليلها 52.76، 47.67 , 49.82 و 39.79 فى المائة حديد و متوسط عائدات حديدية 83.51 ،76.32، 87.63 و 86.83 فى المائة، على التوالى. معدلات تدفق الغذاء فى الفاصل كانت 12.37 ,25.66 , 20.32 و 19.63 جرام لكل دقيقة، على التوالى. هذه النتائج تم تحقيقها عندما كانت شدة التيار الكهربائى للجهاز 0.8 أمبير. و لكن عندما تم تشغيل الجهاز عند تيار مقداره 1.0 امبير تم الحصول على عائدات عالية على حساب الجودة التى نقصت على نحو استثنائى.

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Symbols:

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The following symbols are used in this thesis:

B The magnetic flux density

μ_o The constant of proportionality

H The field intensity

M The intensity of magnetization

S The magnetic susceptibility

μ The relative permeability

F The force on the particle

dH/dl The field gradient

Chapter Three and Four

S The solid density.

DF The density of fluid used.

M_1 The weight of density bottle with stopper.

M_2 The weight of density bottle, stopper and solid material.

M_3 The weight of density bottle, solid material and used liquid.

M_4 The weight of density bottle, stopper and used liquid.

F The weight of the feed

C The weight of concentrate

T The weight of tailings

f The assay of feed

c The assay of concentrate

t The assay of tailings

V_R The variance for the recovery

Vf The variance for the feed

Vc The variance for the concentrate

Vt The variance for the tailings

$\bar{f}$ The mean of the total assay of the feed for the particles coarser than 53 µm

f The total assay of the feed for the particles coarser than 53 µm

n The number of the measurement

SDf The standard deviation of the total assay of the feed

The relative standard deviation of the total assay of the feed RSDf

SDc The standard deviation of the total assays of the concentrates.

SDt The standard deviation of the total assays of the tailings.

$\bar{c}$ and $\bar{t}$ The mean of the total assays of the concentrates and tailing, respectively, for the particles coarser than 53 µm.

RSDc and RSDt the relative standard deviation of the total Assays of the concentrates and the tailings for the particles coarser than 53 µm, respectively.

t Statistical factor

$\bar{x}$ The mean

x_0 The real value

µ The true mean

ε_i The absolute error of measurement i

e_i The algebraic difference between the mean and the ith measurement.

xi The ith measurement

f1 The assay of the particles coarser than 53 um sieve resulted after washing process.

f2 The assay of the particles coarser than 53 um sieve resulted after electromagnetic separation process for Group 1.

f3 The assay of the particles coarser than 53 um sieve resulted after electromagnetic separation process for Group 2.

CS1.n The concentrates of sample (1), for size fraction (n).

TS1.n The tailings of sample (1), for size fraction (n).

CS2.n The concentrates of sample (2), for size fraction (n).

TS2.n The tailings of sample (1), for size fraction (n).

S1.n The feed of sample (1), for size fraction (n)

S2.n The feed of sample (2), for size fraction (n).

CS3.n The concentrates of sample (3), for size fraction (n).

TS3.n The tailings of sample (3), for size fraction (n).

CS4.n The Concentrates of sample (4), for size fraction (n).

TS4.n The tailings of sample (4), for size fraction (n).

S3.n The feed of sample (3), for size fraction (n).

S4.n The feed of sample (4), for size fraction (n)

Chapter One

Introduction and Objectives

1.1. Introduction:

Iron ores are the most commonly used and vital to the modern industry as well as the cheapest of the industrial raw materials. Iron and steel are become recently "key" metals and a country's production of steel is a useful measure of its industrial strength. Although it is not known exactly when or where iron was first employed, or how it was discovered, the jet engines and space-rockets of today are made of special steels developed as the result of experience, which began in the pre-historic times.

Until the 18^{th} century, however, iron smelting remained a small-scale industry using charcoal as fuel for its furnaces. Rock coals (natural coals) were too smoky and too easily crushed for use in smelting furnaces.

In the 1730 an efficient method of turning coal into hard, smokeless coke was developed. Thereafter the industries developed chiefly where iron ores and coking coals were found close together and where was a demand for iron and steel. Pig iron, is a direct solidification of the molten iron, contains about 7% carbon and smaller amount of other impurities such as phosphorus, sulfur, manganese, vanadium, chromium, nickel, tungsten, molybdenum and silica and silicate materials.

To refine Pig iron into steel, all these impurities and most of the carbon impurities must be removed. In order to produce Pig iron having good qualities for the production of steel and its alloyes. Iron ore, coal and limestone should be then beneficiated, if these impurities are present in significant amounts with these materials.

Three chief methods are being employed for upgrading and processing of the iron ores which are the main component of the feed of iron smelting furnaces, the magnetic, the heavy media and flotation methods. Dressing iron ores by magnetic separation began as early as the 18^{th} century, while the heavy media method was introduced into the iron ores processing plants at the beginning of the 19^{th} century. After froth flotation was well established it was introduced into the iron ores plants in the mid of the 20^{th} century.

Rarely one can find one of these methods is used alone, however, two of them or the three methods are used together in the iron ores dressing plants. However, in the arid regions or when the water is dear froth flotation and heavy media processes may be excluded from

the iron ores flow sheets. The characters of the ores are also playing a very important role in the selection of dressing flow sheet to upgrade the ores.

The Sudanese country has several minerals resources chromium, manganese, uranium, copper and very rich in gold. The iron ores might occupy the top of the Sudanese mineral resources. Iron ores are wide distributed in many regions. These ore deposits are found in Karora, Fodikwan, Sofaya, Abu Tulu, Karnoi, Bayuda Desert, Wadi Halfa and El Bjrawiya, however, the soils and sands of Sudan pronounce that there are many other locations would have iron ore deposits which require very extensive exploration and prospecting programs.

Unfortunately, almost iron ores in Sudan of low grade ores which require to be upgraded before their direction towards the industry of steel and iron and the world markets.

The present thesis involves in the beneficiation of the iron ores of El Bajrawiya. The iron ores in this area outcropping in many places. The outcrops of six types of ores appeared there. From these occurrences of ores, samples were collected from the trenches made for the evaluation of the ore reserves of El Bajrawiya.

The materials of these samples were prepared by crushing, grinding, screening and washing in order to liberate the iron oxides grains from the cement materials and silica grains associated with.

As above-mentioned dry and wet beneficiation, processes could be employed to enrich the iron ore oxides. The lack of the apparatus within the mining department for the treatment of minerals has led to process the iron ores of El Bajrawiya by the dry method. The magnetic separation method was employed for the treatment of the iron ores of El Bajrawiya.

This work was carried out by a high intensity magnetic separator (Magnetic Barrier Laboratory Separator).

Since only this apparatus is available within the Central Petroleum Laboratories, to carry out the experimental work. The coarse dry iron ore materials (+53μm) were processed at 0.8 and 1.0 ampere to give two different magnetic fields. The flow rate of the feed into the magnetic field was also varied for different size fractions resulted from the liberation of iron oxide grains as feed for the magnetic apparatus, by grinding and washing processes.

The four produced coarse feed mixtures have an average content ranges from 14.56 to 50.79 % Fe. The separation process gave concentrates have an average content ranges from 39.79 to 52.76 % Fe, and recoveries of iron in the concentrates have an average ranges from 76.32 to 87.63 % , when the strength of the electric current was 0.8 ampere.

However, when the ampere for the magnetic field was increased to 1.0 ampere, the iron of the concentrates were decreased to give ranges from 37.54 to 51.46 % Fe , while the, recoveries of iron in the concentrates were higher, ranging from 91.40 to 95.18 % .

1.2. Objectives:

The objectives of the present research were:

(i) Determination of the qualitative mineralogical composition of the Bajrawiya iron ores.

(ii) Determination of the content of iron in the ore.

(iii) Determination of the distribution of iron in the ore by sizing and assaying tests.

(iv) Determination of the aggregation of the iron . (Microscopic examination and sizing-sorting assaying tests).

(v) Study the existing flow-sheets for the treatment of similar ores.

(vi) Device of a tentative flow-sheet for the laboratory work procedure.

(vii) Study the amenability of the magnetic separation process for the treatment of El Bajrawiya iron ores.

Chapter Two

Literature review

2.1. Introduction:

The literature review being reported in the present thesis includes two parts. Part one will treat the geology of the iron ore deposits of El Bajrawiya, however, part two will review the literature on the works are made to beneficiate the iron ores world wide by magnetic separation method. In addition, the different magnetic machines and systems were included in this part of literature review.

Part one

2.2. Geology of the Iron Ore Deposits:

Systematic prospecting for iron ores in Sudan started in 1820 (s) when Mohamed Ali Pasha became interested in the samples of iron sent from the lateralized Nubian Sandstone at Kordofan Province, Southern Sudan. Since that time, several attempts of iron prospecting and evaluation for different localities had been done but almost all ventures came to nothing. A concise review of these ventures with a list of iron minerals localities in Sudan is given by Whiteman (1971), (Table 2.1). In 1911, Dunn has classified the iron ores of Sudan into three types:

(i) Solid iron ore deposits being unrelated to climatic conditions.

(ii) Surface ore deposits belonging to humid equatorial region, being south of Latitude 10° N (e.g. the lateralized ferruginous soils of Bahr El Ghazal and Upper Nile Province).

(iii) Ferricrete ore deposits belonging to arid regions situated north of Latitude 10° N.

The third type is the concerned El Bajrawiya iron ores, which covers extensive areas, just north of El Bajrawiya Pyramids, north of Shendi, the Nile State. The Bajrawiya iron ores was first reported by Germann *et al* (1990), without a detailed geological analysis and economic evaluation. In 2006, one of the Sudanese Pioneer geologists Abdel Mageed, has visited El Bajrawiya area, and collected spot samples of the different iron ores deposits. The positive chemical analysis of the tested samples encouraged Abdel Mageed to start the

spark of an extensive geological studies and economic evaluation for El Bajrawiya iron ores deposits. Based on his recommendation, a joint team of geologists from HASSA-

Table 2.1 List of iron localities in Sudan, Whiteman, 1971

	Name	Coordinates	Sudan 1:250 000 sheet no.	Comments
Northern Jubal el Bahr el Ahmer				
118.	Fodikwan	21° 44′N 36° 42′E	Dungunab 36–I	Working mine
119.	Hashaseit		—	
120.	Sufaya or Sufaia area	21° 17½′N 36° 16½′E	Dungunab 36–I	Working mine
121.	Yaweikurar		Dungunab 36–I	
122.	Ogra		Mohammed Qôl	
123.	Salala	21° 17′N 36° 11′E	Dungunab 36–I	Associated with Salala ring complex
Nile Valley, Atbara region				
124.	Jebel Hadid, Sudan Portland Cement Quarry, Atbara	17° 38′N 33° 49′E	Atbara 45–K	Replacement deposit in marbles in Basement Complex
Nile Valley, Wadi Halfa region				
125.	Wadi Halfa area		Wadi Halfa 35–I	Sedimentary ore in Nubian Sandstone Formation
126.	Station No. 7	20°–22° N 32°–34° 30′E	J. Murrat	Irregular iron deposits associated with Basement limestones and Nubian Formation
Kordofan				
127.	Jebel Abu Tulu	11° 41′N 28° 40′E	Kadugli 65–D	
Blue Nile Province				
128.	Ingessana Hills Qala en Nahl	—	Er Roseires 66–D	Small magnetite deposits
129.	Blue Nile, near Roseires	—	Er Roseires 66–D	Details unknown, minor occurrence
130.	Jebel Terru	12° 33′N 33° 12′E	Kerkoj 55–O	Magnetite
Southern Sudan				
131.	Various extensive laterite and lateritic deposits			Central and southern Sudan, mainly in Bahr el Ghazal, Equatoria, Upper Nile, Kordofan and Darfur Provinces

mining company and GRAS has carried out detailed geological studies comprising geological setting, sampling, chemical analysis, and then integrative evaluation and estimation of the geological reserve. The final report of these studies concluded that the grade of El Bajrawiya ore is good enough to install steel industry with an estimated geological reserve of about 4 Milliard ton at Fe % ~ 30 for an area of about 2354.6 km², (Abdel Rahman *et al*, 2006). As an extension of the previous evaluation project, this phase represents a detailed investigations including a network of trenches, pits and boreholes with detailed description and sampling for a selected sector of El Bajrawiya area (Plate 2.1), Helba, et al, (2008), Abdelmageed , et al, (2008).

2.2.1. Geological Setting of the Ores:

The study of El Bajrawiya area shows it forms a semicircular plateau or precisely a cuesta has the gentle slope N. NE (Plate 2.2). Its scarps, particularly, the southern one have an irregular outline with southward promontories alternating with northward recessions due to intensive erosion via the northern branches of Wadi El Hawad that debouches its load into the Nile River at Kabushiya Village. The top of the cuesta is nearly flat with isolated low relief hill rocks and ridges, of which the most prominent one is that lies in the central part of the cuesta and is assigned by the name "Ghorabi" Inselberg (Plate 2.2).The flat top is formed essentially of ferruginous sandstone and iron ores accumulations being in many places dissected, eroded and re-deposited as wadi fill and terraces by Quaternary wades., Helba, et al, (2008), Abdelmageed , et al, (2008). The Bajrawyia area lies on the eastern side of the Nile Valley at approximately 40 km south of Atbara Town and 40 km north of Shendi town, Nile State, and bounded from east by Kasala State border. It is located between Latitudes 16° 50'-17° 21', and Longitude 33° 42' -34° 15'.

(i) Stratigraphic Setting:

The rocks forming El Bajrawyia area, and those hosting the concerned iron ores are entirely siliciclastics consisting mostly of quartz sandstone with subordinate kaolinitic / ferruginous mudstone. These siliciclastics belong to the widely distributed Upper Cretaceous Nubian Sandstone Formation, particularly to its upper member "Quartz Sandstone", Kheiralla, 1966. This member is raised to a formational rank by Whiteman (1970) who named it by the formal term "Shendi Formation", and assigned the area northeast of Kabushyia-Um Ali as the type locality (Plate 2.2). At that type area, the Shendi Formation ranges, in the

preserved exposed thickness, between 30 to 50 m. Its base conceals below the surface, and it is unconformable overlain by remnants of the Tertiary Hudi Chert Formation. The

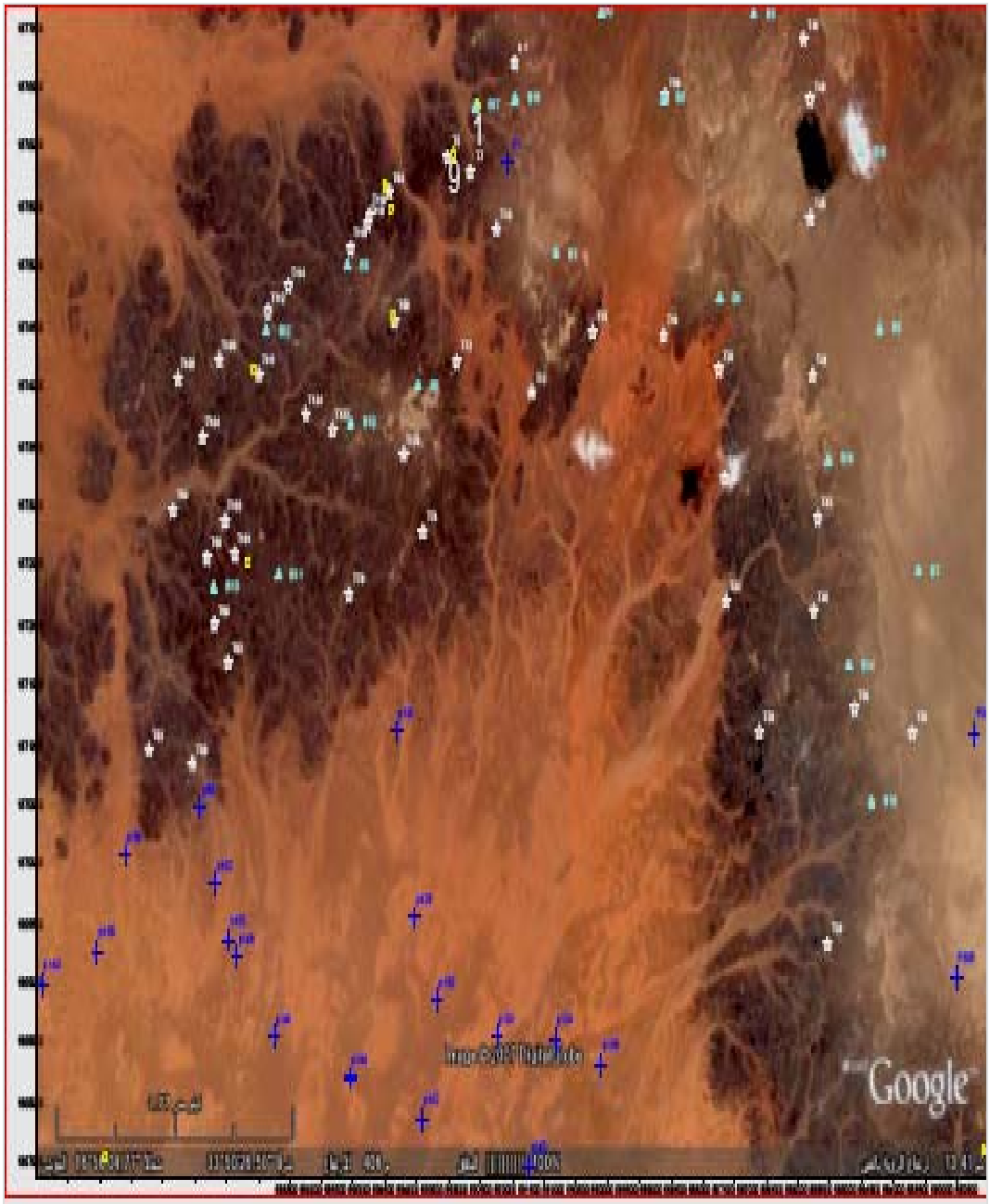

Plate 2.1 Satellite image of the Sector 1 of El Bajrawiya Cuesta showing the locations of the studied trenches, pits, surface sections and boreholes

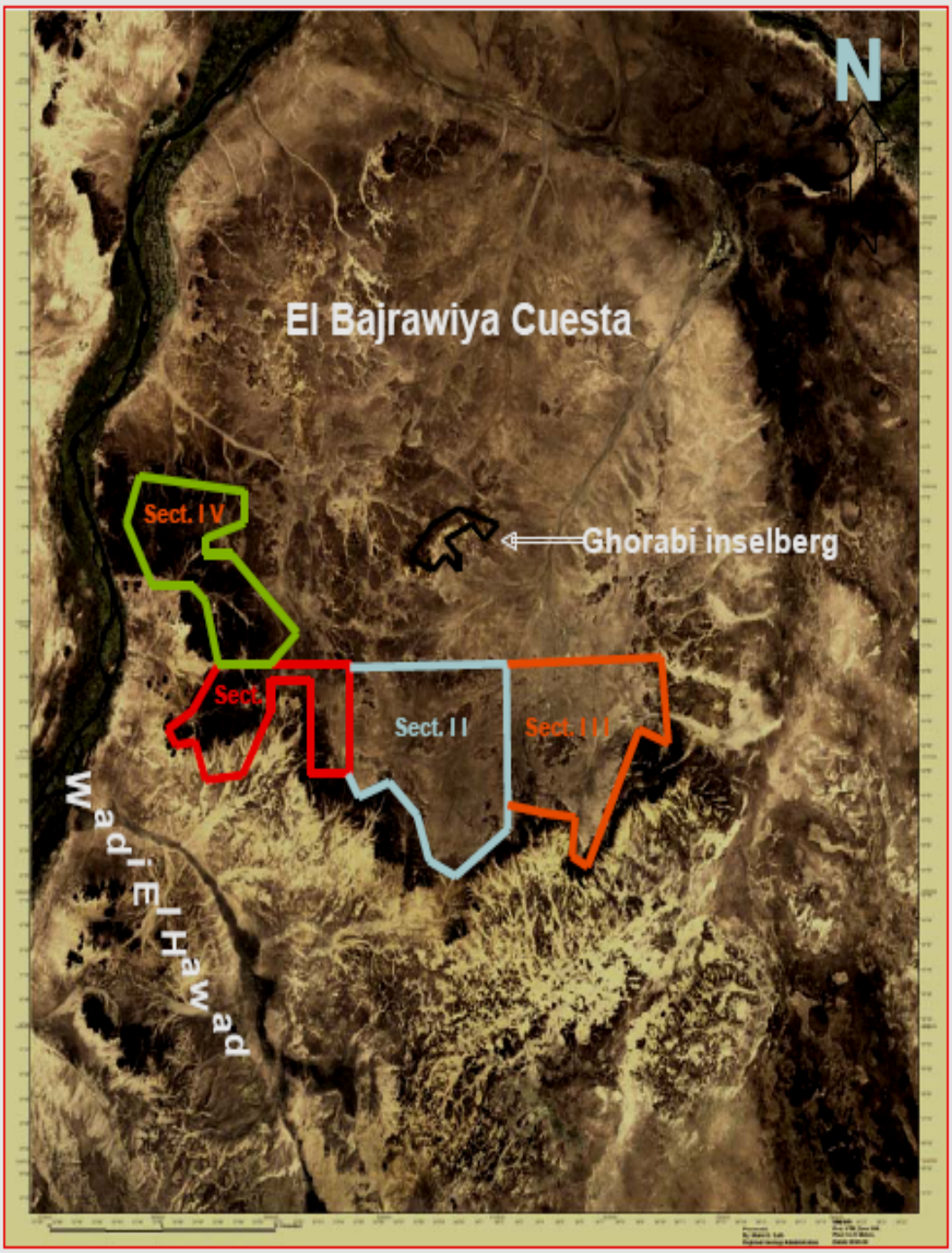

Plate 2.2. Satellite Image of El Bajrawiya Cuesta and the location of the current studying Sector I, and the other sectors to be evaluated in the progress

formation is composed mainly of pale yellow to reddish colored quartz sandstone with subordinate micaceous sandstone. This sandstone is almost friable to weakly consolidated, however, hard iron ores or silica-cemented sandstone also exists capping the hills of the area. The Shendi sandstone is well-sorted, fine to medium-grained, and well-bedded displaying mostly tabular cross-stratification, ripple marks and ribs and furrow structures, which indicate a deposition by water currents that were flowing from SE (Kheiralla (1966), (Plate 2.3).

According to Andrew (1943) and Whiteman (1971), the Shendi Formation in Shendi district is iron rich containing beds of iron minerals that were worked in Meiotic Time, but being now of little economic value, Helba, et al, (2008), Abdelmageed , et al, (2008).

(ii) Shendi Formation (Early Senonian):

The Shendi Formation is easily divisible into three main stratigraphic units (members) being from base to top: lower cross-bedded sandstone; middle flat-bedded sandstone, and upper iron minerals-bearing sandstone (Table 2.2), Helba, et al, (2008), Abdelmageed , et al, (2008).

(iii) Shendi Iron Minerals:

The detailed stratigraphic investigation on Shendi Formation via intensive surface sections and trenches revealed three main intervals of iron minerals deposited, in addition to the reworked iron minerals gravels in the Quaternary wadi fill deposits (Table 2.2). The three main intervals are assigned, from base to top by; lower iron minerals unit (Panorama oolitic iron minerals); middle iron minerals unit (Bog. I iron minerals), and upper iron minerals unit (Bog. II iron minerals), Helba, et al, (2008), Abdelmageed , et al, (2008).

Also Helba, et al, (2008), Abdelmageed , et al, (2008), stated in their prefinal report that :-

1.Stratiform and strata bound ironstones and ferruginous sandstones occur in El Bajrawiya area, which stands as a semicircular plateau or cuesta dipping gently due N. NE, and occupies an area of about 3600 Km².

2.More than 60 representative samples are thin-sectioned and microscopically inspected to determine the macro fabric type, mineralogical composition, micro fabric type and texture as well as the par genetic digenesis and origin.

3.The detailed stratigraphic investigation on sector I revealed that:

a- The rocks forming El Bajrawiya area that hosting the iron minerals accumulations are entirely siliciclasics belonging to the Upper Cretaceous Nubian Sandstone, particularly its upper unit "Shendi Formation". The base of Shendi Formation is unexposed and is unconfor. The mudly overlain by remnant of the Eocene Hudi Chert Formation.

b- Shendi Formation in sector I hosts three main units or intervals of iron minerals, in addition to the reworked iron minerals gravels in the Quaternary wadi fill deposits. The three main intervals are assigned from base to top by:

(i) Lower iron minerals unit (Panorama ironstone)

(ii) Middle iron minerals unit (Bog I)

(iii) Upper iron minerals unit (Bog II)

c- The lower (Panorama) iron minerals unit is the thickest and the highest iron grade ore deposits in Shendi Formation. It measures about 5m thick of high grade Oolitic ironstone without intercalation of ferruginous sediments. Its Fe % ranges from 47.23-56.96, but unfortunately this unit is met with a restricted geographic extension covering an area of about 0.5 Km^2. Laterally from this area, the Oolitic iron minerals pinches and intermingles with ferruginous sandstone. The facies characteristics of this ironstone unit suggest that it was accumulated as channel fill deposits.

d- The middle iron minerals unit (Bog. I) exists in the upper part of Shendi Formation. It outcrops only along the western and southern scarps of sector I, northward and eastward, from the scarps, the unit conceals below the Bajrawiya plateau surface with an overburden ranges from 0.0 up to 8m thick of ferruginous sandstone. The BogI unit attains a preserved thickness ranging between 1 to 2 m. Its greatest thickness reaching to 2.35 m, which is reported in trench No7. Its content of Fe % ranges from 20 % up to 56 %.

e- The upper iron minerals unit (Bog II) constitutes the cap of EI Bajrawiya plateau with a preserved thickness ranging between 1 to 3 m, and nearly without overburden. Its thickest outcrop is measured from trench no.82 attaining about 4 m. This unit of iron minerals is characterized by red to reddish brown color, abundant root burrows and ferruginized wood stems. In almost all studied trenches, the bog II unit is divided into two intervals; the lower one (1 to 2 m, thick) consisting of medium- to thick- beds of sandy mud-iron minerals grading upward to Oolitic iron deposits. The upper interval is formed of cross-bedded ferruginous sandstone (0.3 to 1.5 m, thick) grading upwards to heavily bioturbated sandy mud-iron minerals (0.5 to 1.5 m., thick), with Fe % ranging from 25 to 40 %. This topmost

unit is well developed in Ghorabi Inselberg where it forms its dark ferruginous duricrust cap.

f- Although the cumulative thickness of Bog I and Bog II is relatively small (2 to 4 m, thick), they have a wide geographic extension without a thick overburden that may rise their economic potentiality. However, that geographic extension must be a tested and documented via intense drilling and chemical analysis.

g- The iron minerals of both Bog-I and Bog-II are thin to thick-bedded, internally massive to bioturbated and are almost brecciaed into ill-sorted rubbles. They have higher silica content, and almost traces of Mn-, Mg-, Na-, Ca-, Ph-, and Ba- oxides and low percentage of Al_2O_3 and TiO_2.

4.The detailed microscopic inspection of the representative iron mineral samples revealed that the iron deposit of El Bajrawiya area in sector I comprises 5 main iron minerals types including, in descending order of abundance:

a- Bioturbated silty / sandy mud-iron deposit (type A1)

b- Sandy mud-iron deposit (type A2)

c- Para-autochthonous ooidal / peloidal pack-iron deposit (B3)

d- Sandy autochthonous ooidal / peloidal pack-iron deposit (B2).

e- Autochthonous ooidal / peloidal sandy mud-iron deposit (B1)

In types (A1) and (A2), the iron exists as either cryptocrystalline mud matrix or as authigenic hematite and goethite crystallites that invariably coat, corrode and replace completely the quartz grains. The rock of both types is fine to very fine-grained, moderately to poorly sorted and matrix supported.

In types (B1, B2 and B3), the iron minerals occur as ferriferous peloids and ooids and recrystallized matrix, which also digests and replaces the quartz grains. The rock is medium to coarse-grained, moderately- to well- sorted, and grain-supported.

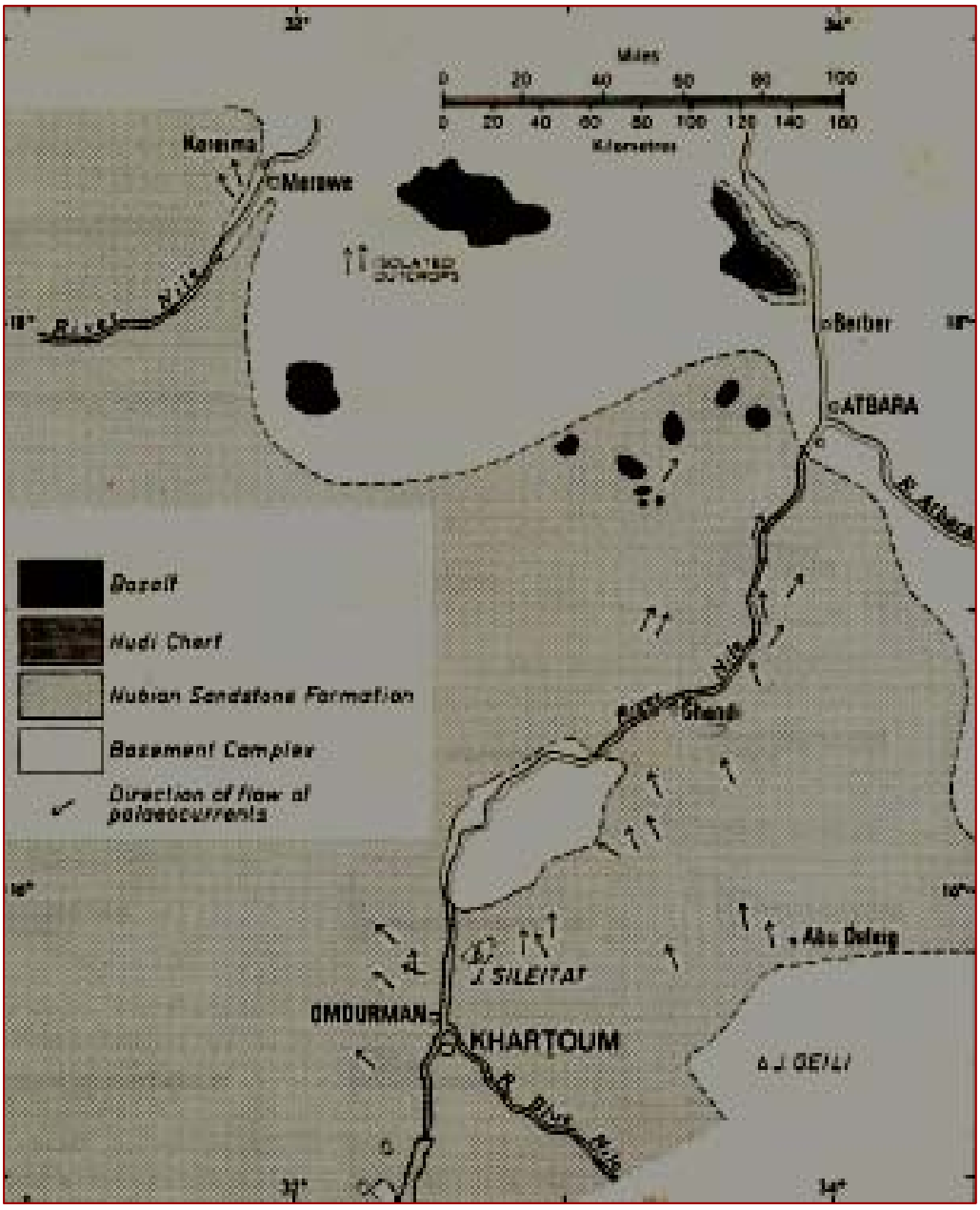

Plate 2.3. Shendi Formation; pale current direction map, Khartoum Shendi district, after Kheiralla, 1966

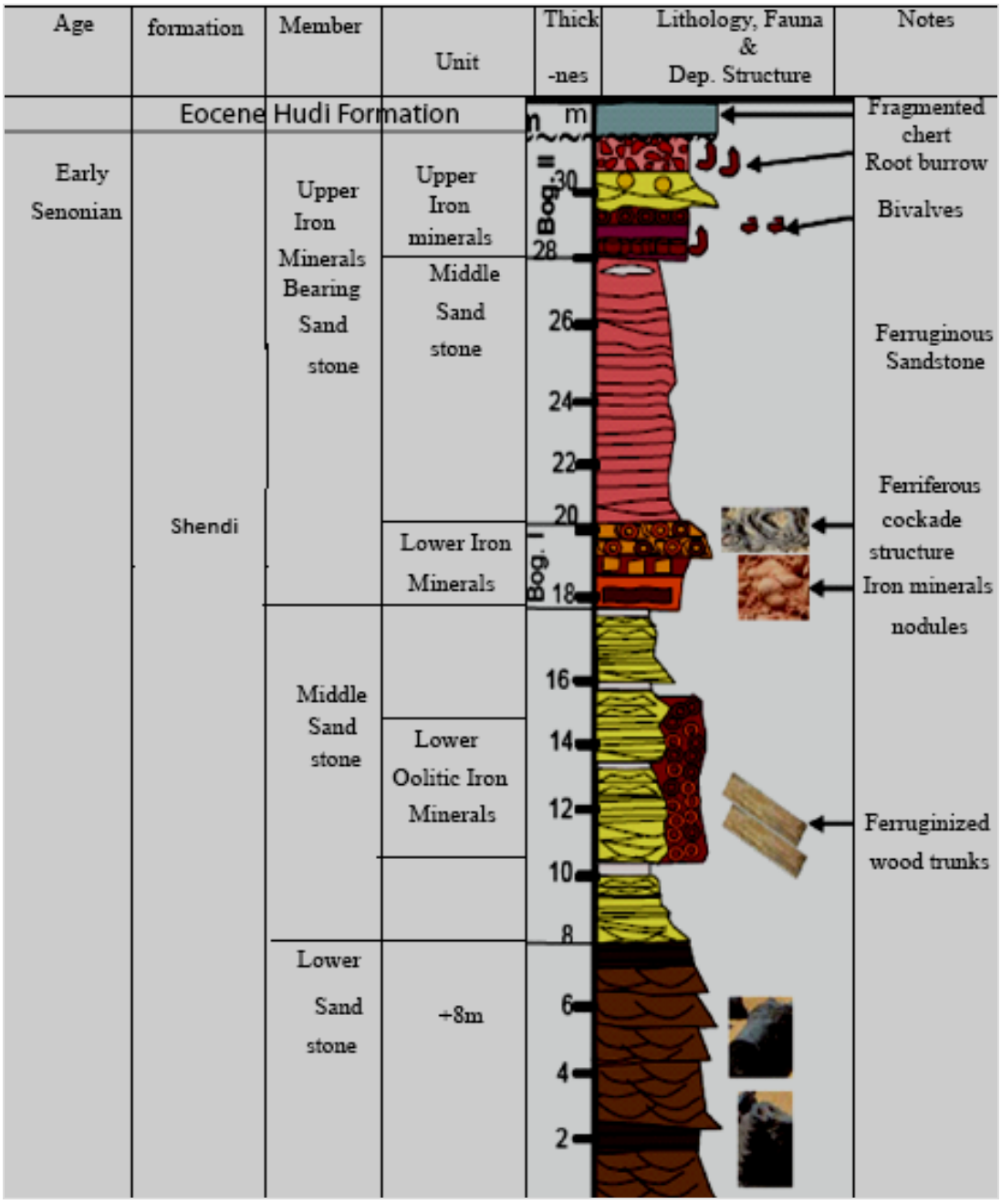

Table 2.2. Compiled stratigraphic section representing the upper cretaceous Shandi Formation and its intervals of iron minerals in the study sector, SW El Bjrawiya plateau

Part Two

2.3. Mineral Separation and its Importance:

2.3.1. Introduction

"As-mined" or "run-of-mine" ore consists to some extent materials (gangue), these materials should be removed from the ore before it being sent into the blast furnace for smelting. Mineral processing, sometimes called ore dressing, mineral dressing or milling, follows mining and prepares the ore for extraction of the valuable metal in the case of metallic ores, and produces a commercial end product of products such as iron ore and coal. Apart from regulating the size of the ore, it is a process of physically ore physicochemically separating the grains of valuable minerals from the gangue minerals, to produce an enriched portion, or concentrate, which containing most of the valuable minerals, and a discard, or tailings, which containing predominantly the gangue minerals. The importance of mineral processing is today taken for granted, but it is interesting to reflect that less than a century ago, ore concentration was often a crude operation, involving relatively simple gravity and hand-sorting techniques performed by the mining engineers. The twentieth century saw the development of mineral processing as a serious and important professional discipline in its own right, and without physical separation, the concentration of many ores, and particularly the metalliferous ores, would be hopelessly uneconomic (Wills and Atkinson, 1991). It has been predicted, however, that the importance of mineral processing of metallic ores may decline as the physical processes utilized are replaced by the hydro and pyrometallurgical routes used by the extractive metallurgist (Gilchrist, 1989), because higher recoveries are obtained by some chemical methods. This may certainly apply when the useful mineral is very finely disseminated in the ore and adequate liberation from the gangue is not possible, in which case a combination of chemical and mineral processing techniques may be advantageous, as is the case with some highly complex ores containing economic amounts of copper, lead, zinc and precious metals (Gray, 1984; Barbery, 1986). Also new technologies such as direct reduction may allow direct smelting of some ores. However, in the majority of cases, the energy consumed in direct smelting or leaching of low-grade ores would be so enormous as to make the cost prohibitive. Compared with these processes, mineral processing methods are inexpensive, and their use is readily justified on economic grounds. If the ore contains worthwhile amounts of more than one valuable mineral, it is usually the object of mineral

processing to separate them; similarly, if undesirable minerals, which may interfere with subsequent refining processes, are present, it may be necessary to remove these minerals at the separation stage. There are two fundamental operations in mineral processing: namely the release, or *liberation,* of the valuable minerals from their waste gangue minerals, and separation of these values from the gangue, this latter process being known as *concentration.* Liberation of the valuable minerals from the gangue is accomplished by comminution, which involves crushing, and, if necessary, grinding, to such a particle size that the product is a mixture of relatively clean particles of mineral and gangue. Grinding is often the greatest energy consumer, accounting for up to 50% of a concentrator's energy consumption, (Wills and Munn, 2006). As it is this process, which achieves liberation of values from gangue, it is also the process, which is essential for efficient separation of the minerals, and it is often said to be the key to good mineral processing. In order to produce clean concentrates with little contamination with gangue minerals, it is necessary to grind the ore finely enough to liberate the associated metals. Fine grinding, however, increases energy costs, and can lead to the production of very fine untreatable "slime" particles, which may be lost into the tailings. Grinding therefore becomes a compromise between clean (high-grade) concentrates, operating costs and losses of fine minerals. If the ore is low grade, and the minerals have very small grain size and are disseminated through the rock, then grinding energy costs and fines losses or slimes can be high, unless the nature of the minerals is such that a pronounced difference in some property between the minerals and the gangue is available. An intimate knowledge of the mineralogical assembly of the ore is essential if efficient processing is to be carried out. A knowledge not only of the nature of the valuable and gangue minerals but also of the ore "texture" is required. The texture refers to the size, dissemination, association and shape of the minerals within the ore. The processing of minerals should always be considered in the context of the mineralogy of the ore in order to predict grinding and concentration requirements, feasible concentrate grades and potential difficulties of separation (Hausen, 1991; Guerney et al., 2003; Baum et al., 2004). Microscopic analysis of concentrate and tailings products can also yield much valuable information regarding the efficiency of the liberation and concentration processes. It is particularly useful in troubleshooting problems which arise from inadequate liberation. Conventional optical microscopes can be used for the examination of thin and polished sections of mineral samples, and in mineral sands applications the simple binocular microscope is a practical tool. However, it is becoming

increasingly common to utilise the new technologies of automated mineral analysis using scanning electron microscopy, such as the Mineral Liberation Analyser (MLA) (Gu, 2003) and the QEMSCAN (Gottlieb, et. al., 2000).

2.3.2. Summary of the Separation Methods:

The most important physical and physco-chemical methods which are used to concentrate ores are:

(i) Separation based on optical and differences in colors and luster of minerals:

This is often called sorting, which used to be done by hand but is now mostly accomplished by machine and electronic devices.

(ii) Separation based on differences in density between the valuable minerals and gangue rocks:

Gravity concentration, a technology with its roots in antiquity, is based on the differential movement of mineral particles in water due to their different hydraulic properties. The method has recently enjoyed a new lease of life with the development of a range of enhanced gravity concentrating devices. In *dense medium separation* particles sink or float in a dense liquid or (more usually) an artificial dense suspension; it is widely used in coal beneficiation, iron ore and diamond processing, and in the preconcentration of metalliferous ores and also for beneficiation of phosphate rocks.

(iii) Separation utilizing the different surface properties between valuable minerals and gangue rocks particles:

Froth flotation, which is one of the most important method of concentration, is effected by the attachment of the mineral particles to air bubbles within an agitated pulp. By adjusting the "climate" of the pulp by various reagents, it is possible to make the valuable minerals air-avid (aerophilic) and the gangue minerals water-avid (aerophobic). This results in separation by transfer of the valuable minerals to the air bubbles which form the froth floating on the surface of the pulp of ground ore and water.

(vi) Separation dependent on electrical conductivity properties:

High-tension separation can be used to separate conducting minerals from non-conducting minerals. This method is interesting, since theoretically it represents the "universal" concentrating method; almost all minerals show some difference in conductivity and it

should be possible to separate almost any two by this process. However, the method has fairly limited application, and its greatest use is in separating some of the minerals found in heavy sands from beach or stream placers. Minerals must be completely dry and the humidity of the surrounding air must be regulated, since most of the electron movement in dielectrics takes place on the surface and a film of moisture can change the behavior completely. The biggest disadvantage of the method is that the capacity of economically sized units is low and its high cost.

(v) Other Methods:

Chemical methods, such as pyrometallurgy or hydrometallurgy, can be used to alter mineralogy, allowing the low cost mineral processing methods to be applied to refractory ores (Iwasaki and Prasad, 1989). For instance, non-magnetic iron oxides can be roasted in a weakly reducing atmosphere to produce ferromagnetic magnetite. It has also been suggested (Parsonage, 1988) that the magnetic response could be increased without chemically altering the minerals, by the adsorption of fine magnetite particles onto the surfaces of non magnetic minerals in the slurry.

(iv) Separation dependent on magnetic properties:

Low intensity magnetic separators can be used to concentrate ferromagnetic minerals such as magnetite (Fe_3O_4), while high-intensity separators are used to separate paramagnetic minerals from their gangue. Magnetic separation is an important process in the beneficiation of iron ores, and finds application in the treatment of paramagnetic non-ferrous minerals. It is used to remove paramagnetic wolframite ((Fe, Mn) WO_4) and hematite (Fe_2O_3) from tin ores, and has found considerable application in the processing of nonmetallic minerals, such as those found in mineral sand deposits.

In many cases, a combination of two or more of the above techniques may be necessary to concentrate an ore economically. Gravity separation, for instance, is often used to reject a major portion of the gangue, as it is a relatively cheap process, It may not, however, have the selectivity or efficiency to produce the final clean concentrate, Gravity concentrates therefore often need further upgrading by other techniques, such as froth flotation, ores which are very difficult to treat *(*refractory*),* due to fine dissemination of the minerals, complex mineralogy, or both, respond very poorly to the above methods, a classic example is the huge zinc-lead-silver deposit at McArthur River, in Australia. Which was discovered in 1955. It is one of the world's largest zinc-lead deposits comprising measured, indicated

and inferred resources totaling 124 Mt with up to 13% Zn, 6% Pb and 60g/t Ag (in 2003) (Wills and Munn, 2006).

It was suggested in this stage that the treatment of El Bjrawiya iron ores would be by the magnetic separation method. Hence, this method was selected to beneficiate the iron ores of El Bjrawiya. The following sections will discuss this process in detail and review the published works carried out in different countries on the process.

2.4. Magnetic Separation:

Magnetic separators exploit the difference in magnetic properties between the minerals of ores and are used to separate either valuable minerals from non-magnetic gangue, e.g. magnetite from quartz, or magnetic contaminants or other valuable minerals from the non-magnetic values. An example of this is the tin-beating mineral cassiterite, which is often associated with traces of magnetite or wolframite which can be removed by magnetic separators. All materials are affected in some way when placed in a magnetic field, although with most substances the effect is too slight to be detected.

Minerals can be classified into three broad groups, according to whether they are attracted or repelled by a magnet:

(i) Diamagnetics are repelled along the lines of magnetic force to a point where the field intensity is smaller. The forces involved here are very small and diamagnetic substances cannot be concentrated magnetically.

(ii) Paramagnetics are attracted along the lines of magnetic force to points of greater field intensity. Paramagnetic materials can be concentrated in high-intensity magnetic separators. Examples of paramagnetics which are separated in commercial magnetic separators are ilmenite ($FeTiO_3$), rutile (TiO_2), wolframite ($(Fe, Mn)WO_4$), monazite (a phosphate of cerium metals, Ce, La and Li), siderite ($FeCO_3$), pyrrhotite (FeS), chromite ($FeCr_2O_4$), hematite (Fe_2O_3), and manganese minerals. Some elements are themselves paramagnetic, such as Ni, Co, Mn, Cr, Ce, Ti, O, and the Pt group metals, but in most cases the paramagnetic properties of minerals are due to the presence of iron in some ferromagnetic form. Ferromagnetism can be regarded as a special case of paramagnetism, involving very high forces, (Wills and Munn, 2006).

(iii) Ferromagnetic materials have very high susceptibility to magnetic forces and retain some magnetism when removed from the field *(*remanence*).* They can be concentrated in low-intensity magnetic separators and the principal ferromagnetic mineral separated is

magnetite (Fe_3O_4), although hematite (Fe_2O_3) and siderite ($FeCO_3$) can be roasted to produce magnetite and hence give good separation. The removal of "tramp" iron from the run of mine ores can also be regarded as a form of low-intensity magnetic separation. (Taggart, 1927).

The unit of measurement of magnetic flux density or magnetic induction (B) which is the number of lines of force passing through a unit area of material, the Tesla (T). The magnetising force which induces the lines of force through a material is called the field intensity (H), and by convention has the units ampere per meter ($1Am^{-1} = 4\prod \times 10^{-7}T$). The intensity of magnetisation or the magnetization (M A/m) of a material relates to the magnetization induced in the material, and:

$B = \mu_o (H + M)$……2.1

Where μ_o is the constant of proportionality, being the permeability of free space, and having the value of $4\prod \times 10^{-7}$T. m/A. In vacuum, M = 0, and it is extremely low in air, therefore Equation (2.1) becomes as fellows:

$B = \mu_o H$……2.2

so that the value of the field intensity is virtually the same as that of flux density, and the term magnetic field intensity is then often loosely used. However, when dealing with the magnetic field inside mate dais, particularly ferromagnetics that concentrate the lines of force, the value of the induced flux density will be much higher than the field intensity, and it must be clearly specified which term is being referred to. Magnetic susceptibility (S) which is the ratio of the intensity of magnetisation produced in the material to the magnetic field which produces the magnetization. Hence,

$S = M/H$……2.3

Combining Equations 2.1 and 2.3 will give

$B = \mu_o H(1 + S)$

or $B = \mu\ \mu_o H$……2.4

where μ equals $1 + S$, and is a dimensionless number known as the relative permeability.

For paramagnetic materials, S is a small positive constant, however is a negative constant for the diamagnetic materials. Figure (2.1) shows plots of induced magnetisation (M) versus the strength of the external field (H), for paramagnetic (hematite) and diamagnetic (quartz) materials. Both plots show straight line relationships between M and H, in each case the slope representing the magnetic susceptibility (S) of the material, i.e. about 0.01 for hematite and around -0.001 for quartz (Wills and Munn, 2006).

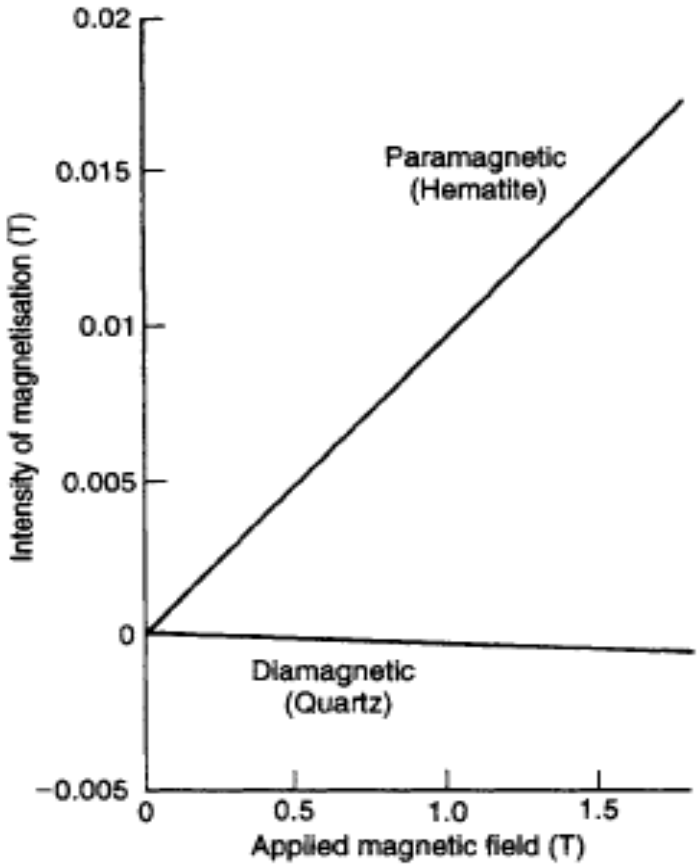

Fig. 2.1 Shows the relationships between intensity of magnetization and applied magnetic field for paramagnetic and Diamagnetic materials, (After Wills and Munn, 2006).

The magnetic susceptibility of a ferromagnetic material is dependent on the magnetic field, decreasing with field strength as the material becomes saturated. Figure (2.2) shows a plot of M versus H for magnetite, showing that at an applied field of 1 T the magnetic susceptibility is about 0.35, and saturation occurs at about 1.5 T. Many high-intensity magnetic separators use iron cores and frames to produce the desired magnetic flux concentrations and field strengths. Iron saturates magnetically between 2 and 2.5 T, and the non-linear ferromagnetic relationship between inducing field strength and magnetisation intensity necessitates the use of very large currents in the energising coils, sometimes up to hundreds of amperes. The capacity of a magnet to lift a particular mineral is dependent not only on the value of the field intensity, but also on the field gradient, i.e. the rate at which the field intensity increases towards the magnet surface. Because paramagnetic minerals have higher magnetic permeabilities than the surrounding media, usually air or water, they

concentrate the lines of force of an external magnetic field. The higher the magnetic susceptibility, the higher is the field density in the particle and the greater is the attraction up the field gradient towards increasing field strength (Wills and Munn, 2006).

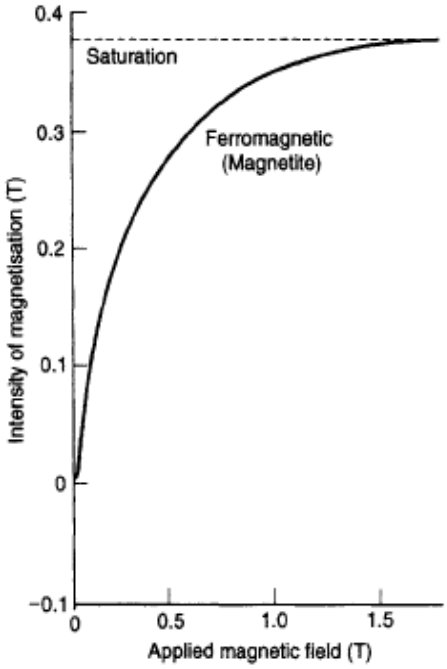

Fig. 2.2 Magnetisation curve for ferromagnetic material

Diamagnetic minerals have lower magnetic susceptibility than their surrounding medium and hence expel the lines of force of the external field. This causes their expulsion in the direction down the gradient of the field towards the decreasing field strength. This negative diamagnetic effect is usually orders of magnitude smaller than the positive paramagnetic attraction (Cohen, 1986).

The following equation shows the force applied or imposed on a particle in a magnetic field

$F \propto H(dH/dl)$…….2.5

where F is the force on the particle, H is the field intensity, and dH/dl is the field gradient. Thus in order to generate a given lifting force, there are an infinite numbers of combinations of field and gradient which will give the same effect. Production of a high field gradient as well as high intensity is therefore an important aspect of separator design (Wills and Munn, 2006).

2.5. Magnetic Separator Design:

Certain elements of design are incorporated in all magnetic separation machines, whether they are low or high intensity, wet or dry. The prime requirement, as has already been mentioned, is the provision of a high-intensity field in which there is a steep field strength gradient. In a field of uniform flux, such as in Plate 2.4 (a), magnetic particles will orient themselves, but will not move along the lines of flux. The most straightforward method for producing a converging field is by providing a V shaped pole above a flat pole, as in Plate 2.4 (b). The tapering of the upper pole concentrates the magnetic flux into a very small area giving the high intensity required. The lower flat pole has the same total magnetic flux distributed over a larger area. Thus, there is a steep field gradient across the gap by virtue of the different intensity levels.

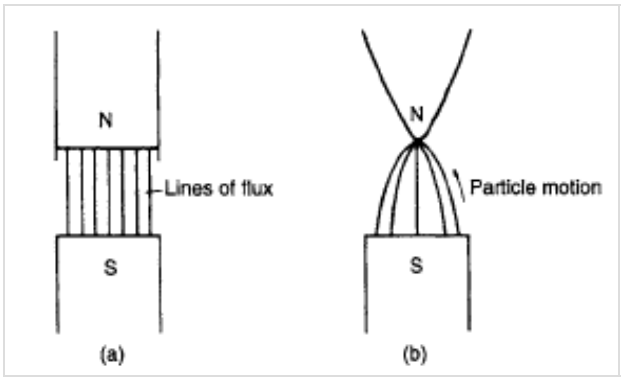

Plate 2.4 (a) Field of uniform flux, (b) converging field

Another method of producing a high field gradient is by using a pole, which is constructed of alternate magnetic and non-magnetic laminations (Plate 2.5). Provision must be incorporated in the separator for regulating the intensity of the magnetic field so as to deal with various types of material. This is easily achieved in electromagnetic separators by varying the current, while with permanent magnets the interpole distance can be varied. Commercial magnetic separators are continuous process machines and separation is carried out on a moving stream of particles passing into and through the magnetic field. Close control of the speed of passage of the particles through the field is essential, which rules out free fall as a means of feeding, (Wills, Munn, 2006)

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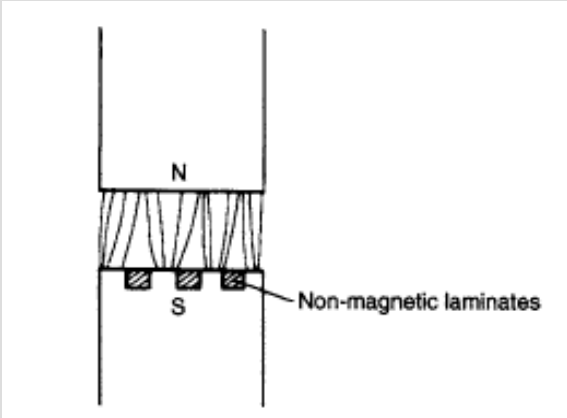

Plate 2.5 Production of field gradient by laminated pole.

Belts or drums are very often used to transport the feed through the field. The introduction into a magnetic field of particles, which are highly susceptible, concentrates the lines of force so that they pass through them, Plate 2.6, (Wills and Munn, 2006).

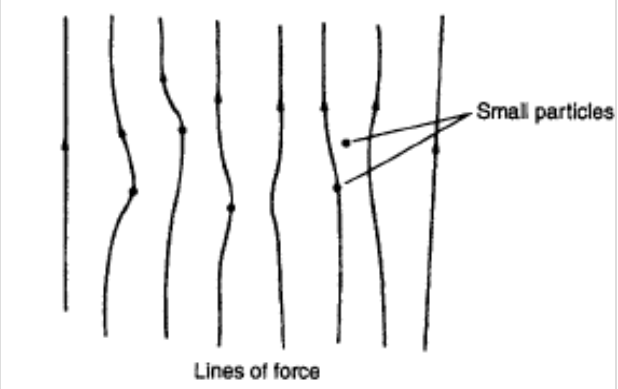

Plate 2.6 Concentration of flux on mineral particles

Since the lines of force converge to the particles, a high field gradient is then produced, which causes the particles themselves to behave as magnets, thus attracting each other, forming flocs or agglomerates of the particles can occur if these particles are very small and highly susceptible and if also the field is intense. This has great importance as these magnetic "flocs" can entrain gangue with and may bridge the gaps between magnetic poles, reducing the efficiency of separation, (Wills and Munn, 2006).

Flocculation is especially serious with dry separating machines operating on fine material. If the ore can be fed through the field in a monolayer, this effect is much less serious, but, of course, the capacity of the machine will be drastically reduced. Flocculation is often

minimised by passing the material through consecutive magnetic fields, which are usually arranged with successive reversal of the polarity. This causes the particles to turn through 180^{o}, each reversal tending to free the entrained gangue particles. The main disadvantage of this method is that flux tends to leak from pole to an other pole, reducing the effective field intensity, (Wills and Munn, 2006).

Provision for collection of the magnetic and non-magnetic fractions must be incorporated into the design of the separator. Rather than the allowance of the magnetics to contact the pole-pieces, which would cause problems of detachment, most separators are designed so that the magnetics are attracted to the pole-pieces, but come into contact with some form of conveying device, which carries them out of the influence of the field, into a bin or a belt. Non-magnetic disposal presents no problems, since they free fall from a conveyor into a bin. This is often being used in practice. Middlings are readily produced by using a more intense field after the removal of the highly magnetic fraction, *(* Wills and Munn, 2006) .

2.6. Types of Magnetic Separators:

Magnetic separators can be classified into low- and high-intensity machines, which may be further classified into dry-feed and wet-feed separators. Low-intensity separators are used to treat ferromagnetic materials and some highly paramagnetic Minerals, (Wills and Munn, 2006) .

2.6.1. Low-Intensity Magnetic Separation:

Dry low-intensity magnetic separation is confined mainly to the concentration of coarse sands, which are strongly magnetic. The process being known as cobbling, and often being carried out in drum separators for particles below the 0.5 cm size, dry separation tends to be replaced by wet methods, which produce much less dust loss and usually a cleaner product. Low-intensity wet separation is now widely used for recovering the magnetite or ferrosilicon particles from magnetic medium in the dense medium separation process, as well as for the concentration of ferromagnetic sands.

Drum separators are the most common machines in current use for cleaning the medium in DMS circuits and are widely used for concentrating finely ground iron ore. They consist essentially of a rotating non-magnetic drum (Figure 2.3) containing three to six stationary magnets of alternating polarity. The Permos separator uses many small magnet blocks, whose direction of magnetization changes in small steps. This is said to generate a very even magnetic field, requiting less magnetic material (Wasmuth and Unkelbach, 1991).

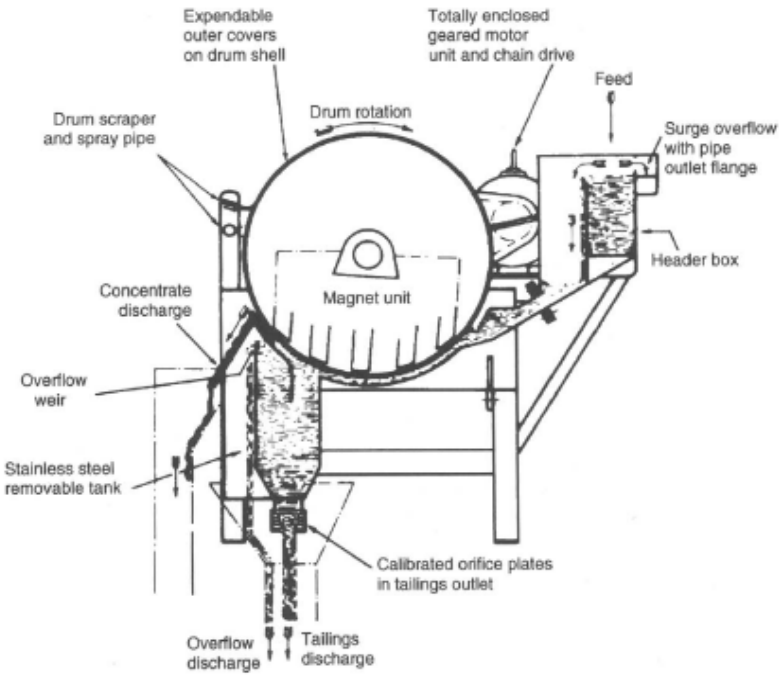

Fig. 2.3 Drum separator

Although initially drum separators employed electromagnets, permanent magnets are used in modem devices, utilising ceramic or rare earth magnetic alloys, which retain their intensity for an indefinite period (Norrgran and Marin, 1994).

Separation is by the "pick-up" principle: the magnetic particles are lifted by the magnets and pinned to the drum and are conveyed out of the field, leaving the gangue materials in the tailings compartment. Water is introduced into the machine to provide a current which keeps the pulp in suspension. Field intensities of up to 0.7 T at the poles surfaces can be obtained in this type of separators. The drum separator shown in Figure 2.3 is the concurrent types, whereby the concentrate is carried forward by the drum and passes through a gap, where it is compressed and dewatered before leaving the separator. This design is most effective for producing an extremely clean magnetic concentrate from relatively coarse materials and is widely used in dense medium recovery systems. This separator which is shown in Figure 2.4, a is a counter-rotation type, where the feed flows in the opposite direction to the rotation. This type is used in roughing operations, where occasional surges in feed must be handled, where magnetic material losses are to be held to a minimum, while an extremely clean concentrate is not required, and when high solids loading is encountered.

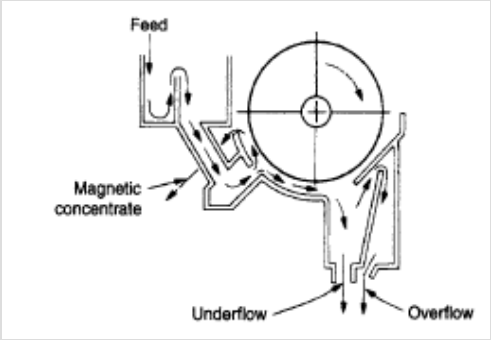

Fig. 2.4, a Counter-rotation drum separator

Figure 2.4, b shows a counter-current separator, where the tailings are forced to travel in the opposite direction to the drum rotation and are discharged into the tailings chute.

This type of separator is designed for finishing operations on relatively fine material, of particle size less than about 250 µm. Drum separators are widely used to treat lowgrade taconite ores, which contain 40-50% Fe, mainly as magnetite, but in some areas with hematite, finely disseminated in bands in hard siliceous rocks. Very fine grinding is necessary to free the iron minerals from the gangue minerals to produce a concentrate that may require pelletising before being fed to the blast furnaces. In a typical flowsheet the ore is ground progressively finer, the primary grind usually being undertaken autogenously, or by rod milling, followed by magnetic separation in drum separators. The magnetic concentrate is reground and again treated in drum separators. This concentrate may be further reground, followed by a third stage of magnetic separation. The tailings from each stage of magnetic separation are either rejected or, in some cases, treated by spiral or cone concentrators to recover some of the scaped hematite particles. At Palabora, the tailings from copper flotation are deslimed, after which the +105 µm material is treated by Sala drum separators to recover 95% of the magnetite at a grade of 62% Fe.The cross-belt separator (Figure 2.5) and disc separators once widely used in the mineral sands industry, particularly for recovering ilmenite from heavy mineral concentrates, which are now considered obsolete. They are being replaced with rare earth roll magnetic separators and rare earth drum magnetic separators (Arvidson, 2001).

Rare earth roll separators use alternate magnetic and non-magnetic laminations which was shown before in Plate 2.5 Feed is carried onto the magnetic roll by means of a thin belt as shown in Figure 2.5, hence there is no bouncing or scattering of particles as they enter the magnetic zone, and they all enter the magnetic zone with the same horizontal velocity. These factors contribute to achieving a sharp separation. Roll speed can be adjusted over a

wide range, allowing the product quality to be "dialled in". Dry rare earth drum separators provide a fan of separated particles which can often be seen as distinct streams (Figure 2.6). The fan can be separated into various grades of magnetic product and a nonmagnetic tailings. In some mineral sands applications, drum separators have been integrated with one or more rare earth rolls, arranged to treat the middlings particles from the drum as shown in Figure 2.6, *(* Wills and Munn, 2006)

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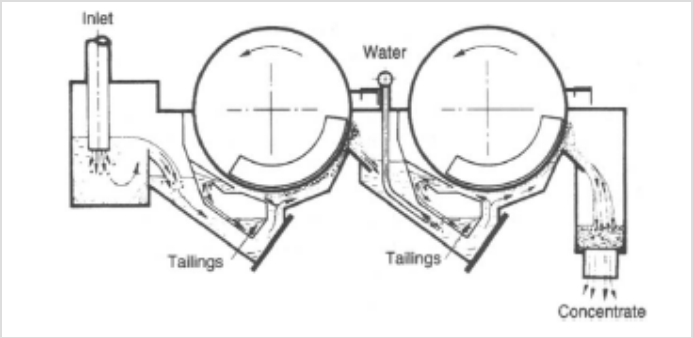


Fig. 2,4, b Counter-current separator

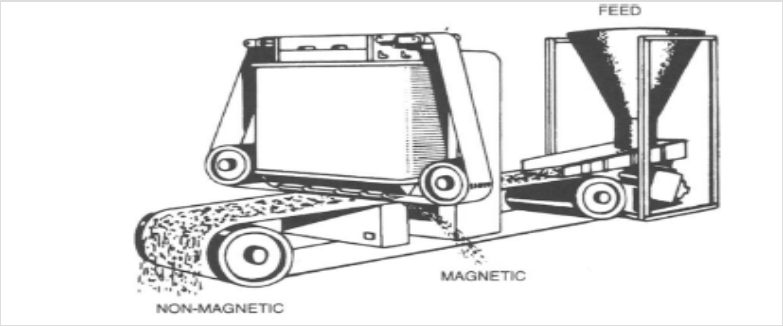


Fig. 2.5 Cross-belt separator

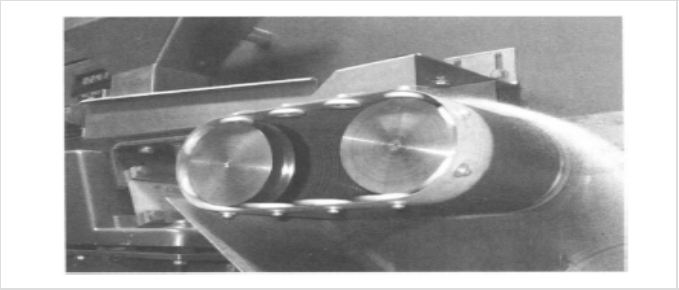


Fig. 2.6 A laboratory dry rare earth drum separator
(Courtesy JKMRC and JKTech Pty Ltd)

2.6.2. High-Intensity Separators:

Very weakly paramagnetic minerals can only be effectively removed from an ore feed if high intensity fields of 2 T and more can be produced (Svoboda, 1994).

Until the 1960s, high-intensity separation was confined solely to dry ore, having been used commercially since about 1908. *Induced roll magnetic separators*, IRMs (Figure 2.7), are widely used to treat beach sands, wolframite, tin ores, glass sands, and phosphate rock. They have also been used to treat feebly magnetic iron ores, principally in Europe. The roll, on to which the ore is fed, is composed of phosphated steel laminates compressed together on a non-magnetic stainless steel shaft. By using two sizes of lamination, differing slightly in outer diameter, the roll is given a serrated profile which promotes the high field intensity and gradient required. Field strengths of up to 2.2 T are attainable in the gap between feed pole and roll. Nonmagnetic particles are thrown off the roll into the tailings compartment, whereas magnetics are gripped, carried out of the influence of the field and deposited into the magnetics compartment. The gap between the feed pole and rotor is adjustable and is usually decreased from pole to pole to take off successively more weakly magnetic products. The setting of the splitter plates cutting into the trajectory of the discharged material is obviously of great importance,.

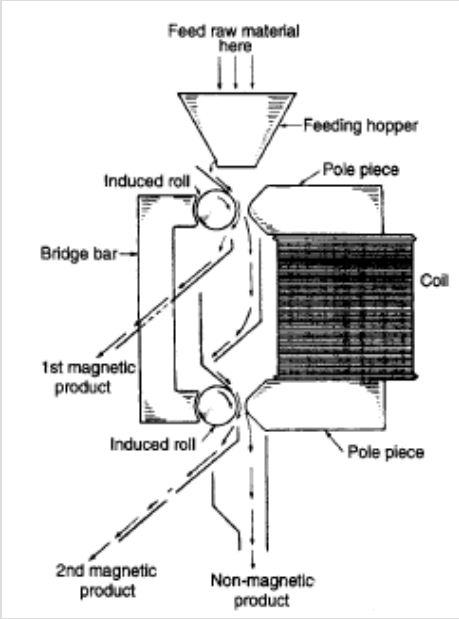


Fig. 2.7 Induced roll separator

In some cases IRMs are now being displaced by the new rare earth drum and roll separators. Dry high-intensity separation is largely limited to ores containing little, if any, material finer than about 75 μm. The effectiveness of separation on such fine material is severely reduced by the effects of air currents, particle-particle adhesion, and particle-rotor adhesion, *(*Wills and Munn, 2006).

Without doubt the greatest advance in the field of magnetic separation was the development of continuous wet high-intensity magnetic separators, WHIMS machines (Lawver and Hopstock, 1974).

These machines reduce the minimum particle size for efficient separation allowing ores to be concentrated magnetically that cannot be concentrated effectively by dry high-intensity methods, because of the fine grinding necessary to ensure complete liberation of the magnetic fraction. In some flowsheets, expensive drying operations can be eliminated by using a wet concentration system. Perhaps the most well-known (WHIMS) machine is the Jones separator*,* the design principle of which is utilised in many other types of wet separator used today. The machine consists of a strong main frame (Figure2.8) made of structural steel. The magnet yokes are welded to this frame, with the electromagnetic coils enclosed in air-cooled cases. The actual separation takes place in the plate boxes which are on the periphery of the one or two rotors attached to the central roller shaft. The feed, which is thoroughly mixed slurry, flows through the separator via fitted pipes and launders into the plate boxes (Figure 2.9), which are grooved to concentrate the magnetic field at the tip of the ridges. Feeding is continuous due to the rotation of the plate boxes on the rotors and the feed points are at the leading edges of the magnetic fields. Each rotor has two symmetrically disposed feed points.

The feebly magnetic particles are held by the plates, whereas the remaining non-magnetic slurry passes straight through the plate boxes and is collected in a launder. Before leaving the field any entrained non-magnetics are washed out by low pressure water and are collected as a middlings product.

When the plate boxes reach a point midway between the two magnetic poles, where the magnetic field is essentially zero, the magnetic particles are washed out with high pressure scour water sprays operating at up to 5 bar (Plate 2.7). Field intensities of over 2 T can be produced in these machines.

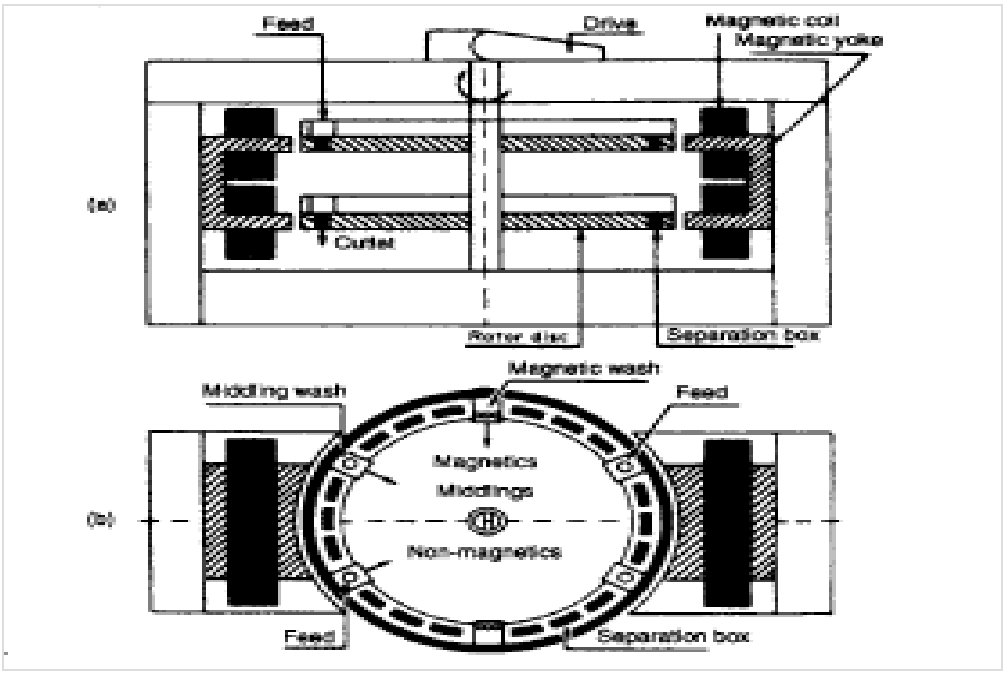


Fig. 2.8 Operating principle of the Jones high-intensity wet magnetic separator in cross-section(a) plan and (b) view

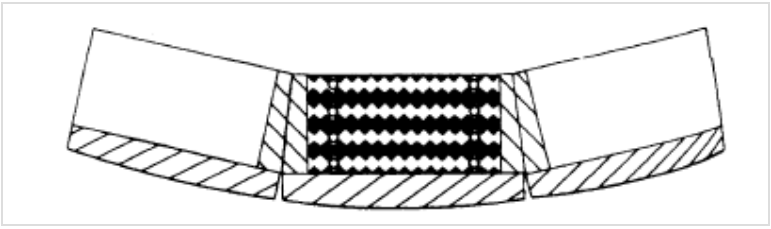


Fig. 2.9 Plan of Jones plate box showing grooved plates and spacer bars

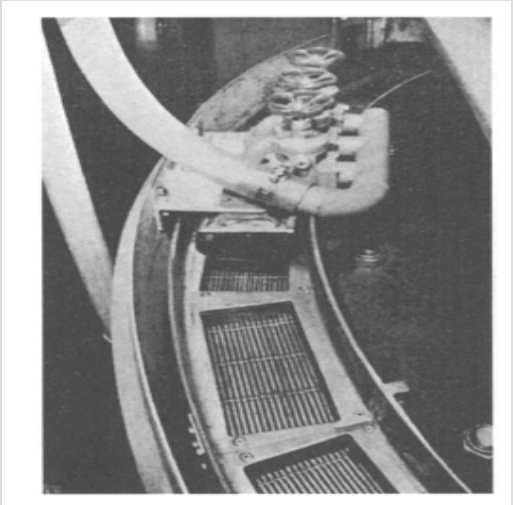


Plate 2.7 Jones separator- magnetic wash

The production of a 1.5 T field requires an electric power consumption in the coils of 16 kW per pole. Of the 4t of water used with every tonne of solids, approximately 90% is recycled. Wet high-intensity magnetic separation has its greatest use in the concentration of low-grade iron ores containing hematite, where they frequently replace flotation methods, although the trend towards magnetic separation has been slow in North America, mainly due to the very high capital cost of such separators. It has been shown that the capital cost of flotation equipment for concentrating weakly magnetic ore is about 20% that of a Jones separator installation, (White, 1978).

Although flotation operating costs are about three times higher, the total cost depends on terms for capital depreciation; over 10 years or longer, the high-intensity magnetic separator may be stile the most attractive process. Additional costs for water treatment may also boost the total for a flotation plant. Plate 2.8 shows a Jones separator in operation on a Brazilian iron ore plant. Various other designs of wet high-intensity separator have been produced, a four-pole machine being manufactured by Boxmag-Rapid Ltd. The plate boxes in this design are an array of magnetic stainless steel "wedge-bars" similar to those used in fine screening (Plate 2.9). In addition to their large-scale application for the recovery of hematite, wet high-intensity separators are now in operation for a wide range of duties, including removal of magnetic impurities from cassiterite concentrates, removal of fine magnetic from asbestos, removal of magnetic impurities from scheelite concentrates, purification of talc, the recovery of wolframite and non-sulphide molybdenum-bearing minerals from flotation tailings, and the treatment of heavy mineral beach sands. They have also been successfully used for the recovery of gold and uranium from cyanidation residues in South Africa (Corrans, 1984).

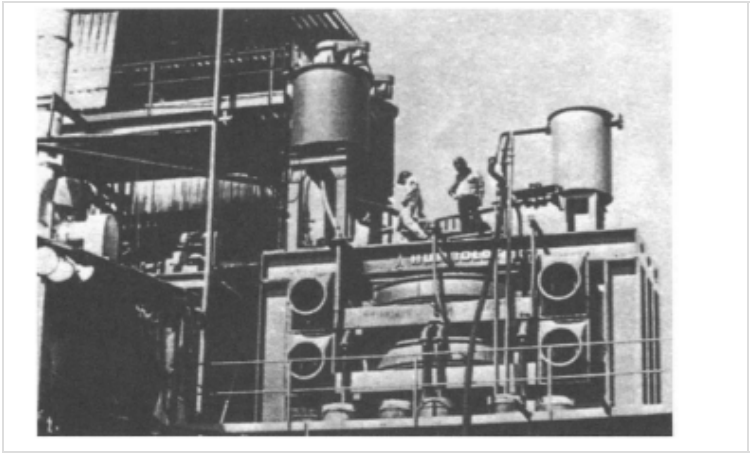


Plate 2.8 Jones separator treating Brazilian hematite ore

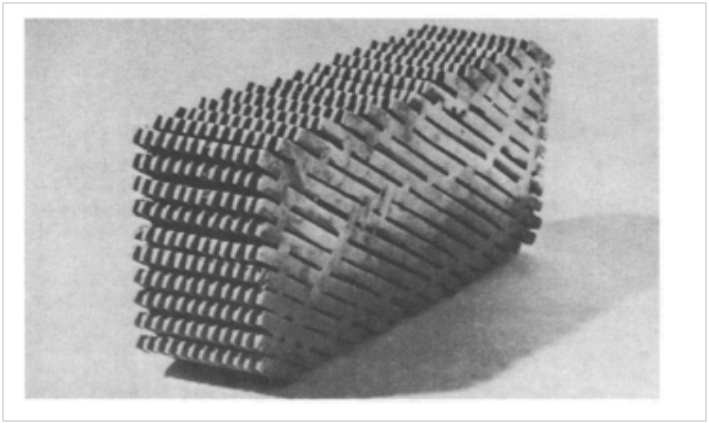


Plate 2.9 Section through Boxmag-Rapid grid assembly

These residues contain some free gold, while some of the fine gold is locked in sulphides, mainly pyrite, and in various silicate minerals. The free gold can be recovered by further cyanidation, while flotation can recover the pyritic gold. Magnetic separation can be used to recover some of the free gold, and much of the silicate-locked gold, due to the presence of iron impurities and coatings. The paramagnetic properties of some sulphide minerals, such as chalcopyrite and marmatite, have been exploited by applying wet high-intensity magnetic separation to augment differential flotation processes commonly used to separate these minerals from less magnetic or non-magnetic sulphides (Tawil and Morales, 1985).

Test work showed that a Chilean copper concentrate could be upgraded from 23.8 to 30.2% Cu, at 87% recovery. This was done by separating the chalcopyrite from pyrite in a field of 2 T. In Cu-Pb separation operations, it was found that chalcopyrite and galena could be effectively separated with field strengths as low as 0.8 T. When the process was applied to the de-coppering of a molybdenite concentrate, it was possible to reduce the copper content from 0.8 to 0.5% with over 97% Mo recovery, *(* Wills and Munn, 2006) .

2.6.3. High-Gradient Magnetic Separators:

In order to separate paramagnetic minerals of extremely low magnetic susceptibility, high magnetic forces must be generated. These forces can be produced by increasing the magnetic field strength, and in conventional high-intensity magnetic separators use is made of the ferromagnetic properties of iron to generate a high B-field (induced field) many hundreds of times greater than the applied H-field, with a minimum consumption of electrical energy. The working field occurs in air-gaps in the magnetic circuit, the disadvantage being that the volume of iron required is many times greater than the gap

volume where separation takes place. The steel plates in a Jones separator, for example, occupy up to 60% of the process volume. Thus high-intensity magnetic separators using conventional iron circuits tend to be very massive and heavy in relation to their capacity. A large separator may contain over 200 t of iron to carry the flux, hence capital and installation costs are extremely high. As iron saturates at around 2-2.5 T, conventional iron circuits are of little value for generating fields above about 2 T. Such fields can only be generated by the use of high H-fields produced in solenoids, but the energy consumption is extremely high and there will be many problems in cooling the solenoid. An alternative is to increase the magnetic force by increasing the value of the magnetic field gradient. Instead of using one large convergent field in the gap of a magnetic circuit, the uniform field of a solenoid is used (Figure 2.10). The core, or working volume, is filled with a matrix of secondary poles, such as ball bearings, or wire wool, the latter filling only about 10% of the working volume. Each secondary pole, due to its high permeability, can produce a maximum field strength of the order of 2 T, but more importantly, each pole produces, in its immediate vicinity, high field gradients of up to 14 T mm $^{-1}$. Thus a multi rode of high gradients across numerous small gaps, centred around each of the secondary poles, is achieved.

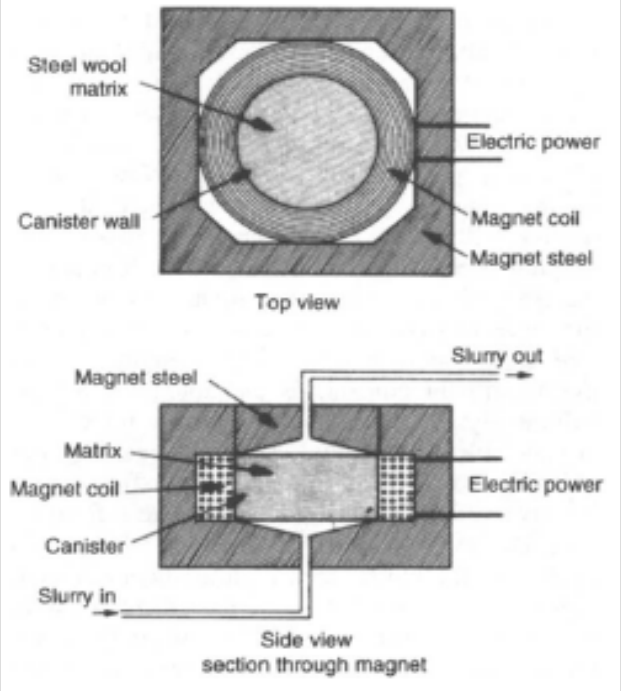


Fig. 2.10 High-gradient magnetic separator

The solenoid can be clad externally with an iron frame to form a continuous return path for the magnetic flux, thus reducing the energy consumption for driving the coil by a factor of about 2. The matrix is held in a canister into which the slurry is fed. As particles are captured, the ability of the matrix to extract particles is reduced. Periodically the magnetic field can be removed and the matrix flushed with water to remove the captured material. An inherent disadvantage of high gradient separators is that an increase in field gradient necessarily reduces the working gap between secondary poles, the magnetic force having only a short reach, usually not more than 1 mm. It is therefore necessary to use gaps of only about 2mm between poles, such that the matrix separators are best suited to the treatment of very fine particles. They are used mainly in the kaolin industry for removing micron-sized particles which contain iron. Several large separators, with the ferromagnetic matrix contained in baskets approximately 2 m in diameter are in commercial use in the United States and in Cornwall, England. They operate with fields of 2 T, and have capacities ranging between 10 and 80t h^{-1} depending on the final clay quality desired. One of the most important factors which will affect coal preparation policy in the future is the environmental issue associated with acid rain and its link with sulphur emissions from fossil fuels. Sulphur occurs in coal in three forms. It is part of the coal substance (organic sulphur), or occurs as the minerals pyrite and marcasite, or as sulphates. The most important factor for the engineer is the pyritic sulphur content, as technology is not yet sufficiently developed to consider the removal of organic sulphur. If pyrite can be liberated by fine comminution to around 1 mm, then froth flotation or gravity methods can be used to remove it from the coal. However, if very fine commiution is necessary to liberate the pyrite, then high-gradient magnetic separation is a possibility. Increased international interest is at present being shown by coal preparation engineers in coal-liquid mixtures as a replacement for conventional hydrocarbon fuels such as diesel oil and natural gas. A typical coal-water mixture consists of pulverised coal of less than 50 microns particle size, and low ash content (2-6%) dispersed in an aqueous slurry, with a pulp density of between 50 and 80% solids. In order to produce these mixtures it is necessary to treat good quality coal by fine grinding and deep cleaning to remove ash and sulphur. High-gradient magnetic separation is capable of removing pyrite from pulverised coal, and much work is currently being performed on a variety of coal types (Lua and Boucher, 1990).

2.6.4. Superconducting Separators:

Undoubtedly, the future developments and applications of magnetic separation in the mineral industry will lie in the use of high magnetic forces. Matrix separators with very high field gradients and multiple small working gaps can draw little advantage from field strengths above the saturation levels of the secondary poles. However, "opengradient" separators, with large working volumes to *deflect* coarser particles at high capacity, rather than *capture* particles, as in high-gradient separators, need to use the highest possible field strengths in order to generate the high magnetic forces required to treat feebly paramagnetic particles. Field strengths in excess of 2 T can only be generated economically by the use of *superconducting magnets* (Kopp, 1991; Watson, 1994).

Certain alloys have the property of presenting no resistance to electric currents at extremely low temperatures. An example is niobium-titanium at 4.2 K, the temperature of liquid helium. Once a current commences to flow through a coil made from such a superconducting material, it will continue to flow without being connected to a power source, and the coil will become, in effect, a permanent magnet. Superconducting magnets can produce extremely intense and uniform magnetic fields, of up to 15 T. The main problem, of course, is in maintaining the extremely low temperatures, and in 1986 a Ba / La / Cu oxide composite was made superconductive at 35 K, promoting a race to prepare ceramic oxides with much higher superconducting temperatures (Malati, 1990). Unfortunately, these materials are of a highly complex crystal structure, making them difficult to fabricate into wires. They also have a low current-carrying capacity, so it is likely that for the foreseeable future superconducting magnets will be made from ductile niobium alloys, embedded in a copper matrix. In 1986 a superconducting high-gradient magnetic separator was designed and built by Eriez Magnetics to process kaolinite clay in the United States (Stefanides, 1986). This machine uses only about 0.007 kW in producing 5 T of flux, the ancillary equipment needed requiting another 20kW. In comparison, a conventional 2T high-gradient separator of similar throughput would need about 250 kW to produce the flux, and at least another 30 kW to cool the magnet windings. The 5T machine is an assembly of concentric components (Figure 2.11). A removable processing canister is installed in a processing chamber located at the centre of the assembly. This is surrounded by a double-walled, vacuum insulated container that accommodates the superconductive magnet's niobium/titanium-tantalum winding, and the liquid helium coolant. A thermal

shield, cooled with liquid nitrogen to77 K, limits radiation into the cryostat. In operation, the supply of slurry is periodically cut off, the magnetic field is shut down, and the canister backwashed with water to clear out accumulated magnetic contaminants.

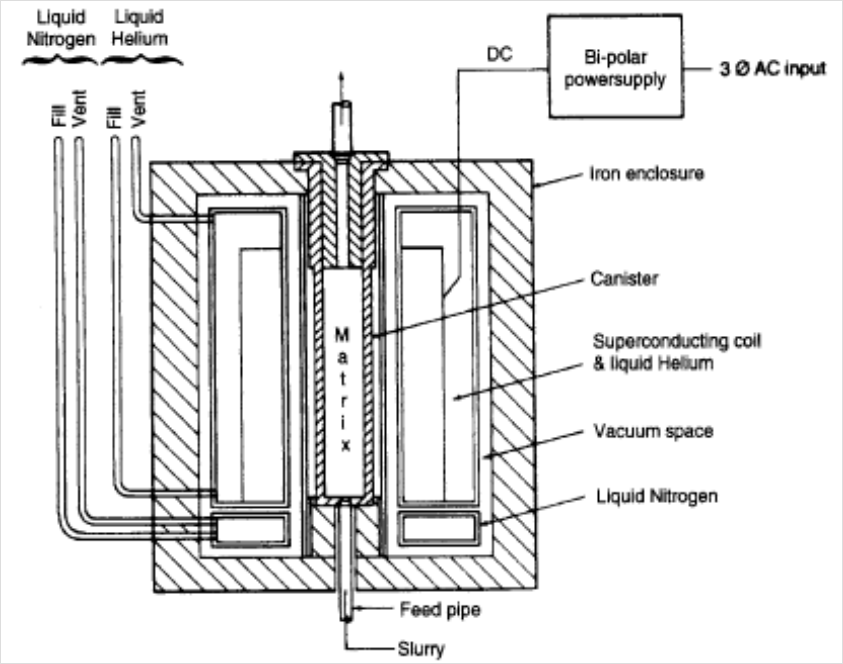


Fig. 2.11 5 T superconducting magnetic separator

An open-gradient drum magnetic separator with a superconducting magnet system has been operating commercially since the late 1980s (Unkelbach and Kellerwessel, 1985; Wasmuth and Unkelbach, 1991) (Figure 2.12).

Although separation is identical to that in conventional drum separators, the magnetic flux density at the drum surface can reach over 4 T, generated by the superconductive magnet assembly within the drum.

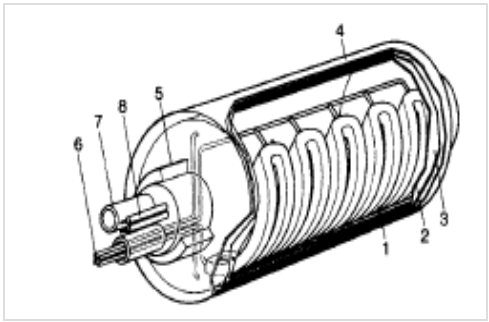


Fig. 2.12 Superconducting drum separator: 1 - magnetic coils, 2 - radiation shield, 3 – vacuum tank, 4 - drum, 5 - plain bearing, 6 - helium supply, 7 - vacuum line, 8 - current supply

2.6.5. The Frantz® Magnetic Barrier Laboratory Separator Model LB-1:

A mixture of diamagnetic and weakly paramagnetic grains undergoes separation in the field of the Magnetic Barrier Laboratory Separator (Model LB-1).

The magnetic system is inclined so that gravity urges particles toward the far side of the chute and down its length. The light colored diamagnetic grains are deflected along the magnetic barrier, while the darker paramagnetic grains pass through it and out of the fields in the channel on the far side of the divider, Plate 2.10 .

Plate 2.10

Separation in the Barrier Field

The stream of particles traveling through the field of the Magnetic Barrier Laboratory separator is split by opposed magnetic and nonmagnetic forces. Magnetic force transverses to field direction deflects particles of selected susceptibility from the paths in which gravity urges them. The process is continuous.

The paths of all material undergoing separation, including the most strongly magnetic components, are parallel to the midplane between the opposed faces of the pole pieces, perpendicular to the flux path. Near the midplane, where the stream of material undergoes separation, force attracting particles toward the pole pieces is negligible, no matter how strongly they are magnetized.

Magnetic energy gradient $\{H*(dH/dx)\}$ in the separating region of the Barrier field has a pattern that may be likened to a packet of thin sheets, with their surfaces aligned with the lengthwise axis of the gap between the pole pieces. Along the length of each sheet magnetic energy gradient transverse to field direction is constant. Across the width of the separating region magnetic energy gradient transverses to field direction rises from sheet to sheet, from low values at the outer fringes of the field to maximum value at a sheet near the center of the separating region. Material is moved by gravity across the field, through the succession of sheets of ascending magnetic force, toward the region of maximum transverse force. Particles of like susceptibility encounter like magnetic force per unit

volume. Particles having susceptibility such that magnetic force opposing their motion exceeds gravitational force are deflected in the vicinity of the sheet of maximum transverse force, while particles having susceptibility that is weaker or of opposite sign pass through it. A component of gravity urges both fractions toward a mechanical divider and out of the field. Conventional separators employ magnetic force aligned with field direction to attract particles toward magnetized collecting surfaces. Magnetic force has maximum value at such a surface and decreases rapidly with distance from the surface. The force experienced by any particle in conventional separators depends on the accident of its position in the field. Particles passing close to a surface may be captured, while particles passing farther away may escape.

The Barrier technology, by providing conditions in which particles of like susceptibility encounter like magnetic force per unit volume, has inherent advantages in selectivity and sensitivity for separating materials according to slight differences in susceptibility.

Superimposed graphs of magnetic field intensity, H, and magnetic energy gradient, $H^{dH}/_{dx,}$ at the midplane x-x', between the opposed pole faces show the characteristics of the barrier field and its location near the outer edge of the gap. The graphs are based on measurements taken at the midplane with maximum current of about 1.8A supplied to the coils. In the katadynamic region, k, energy gradient increases in the direction in which field intensity increases. At the isodynamic plane, i, it reaches maximum value. In the Ana dynamic region, a, energy gradient declines in the direction in which field intensity increases, reaching zero value where field intensity is maximum and uniform, Figure 2.13.

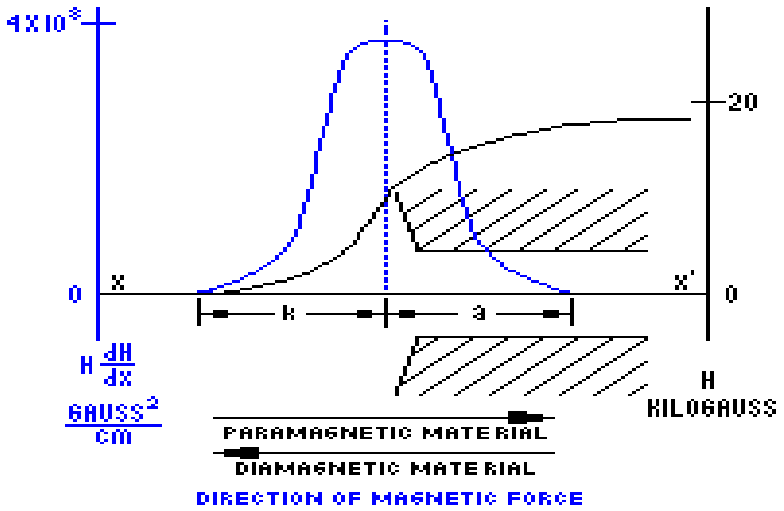


Fig. 2.13

Separation by exploiting paramagnetic susceptibility is shown here. The mixture is fed into the anadynamic region inside the gap of the LB-1. The magnetic system is inclined so that gravitational force urges particles toward the near side of the chute. The darker paramagnetic grains can be seen moving along the magnetic barrier, while the light colored diamagnetic grains pass through it, Plate 2.11 The sample is the same mixture of diamagnetic and weakly paramagnetic grains used for the separation shown in Plate 2.10, (Jack J. Sun, 1980).

Plate 2.11

2.6.6 Conversion of Isodynamics to Barrier Separators:

The Barrier inventions carry forward a branch of magnetic separation technology which began with inventions by Samuel G. Frantz, the founder of the Company. The Frantz Isodynamic Separator, embodying these inventions, became the leading device for separation according to magnetic susceptibility. The inventions are widely acknowledged to have provided the key to selective separation.

The Frantz Isodynamic Separator has a virtually unlimited useful life. The elements of the Magnetic Barrier Laboratory Separator were consequently designed so that all parts can be installed in the base and the magnetic circuit of any Isodynamic Separator to convert it to a Barrier. (The first prototype of the Isodynamic Separator, manufactured in 1935 by Frantz, was in service at Bryn Mawr College in 1989, when it was converted to a Barrier.) The Bundesanstatt for Geowissenschaft und Rohstoffe of Hannover, Germany, converted one Isodynamic Separator to a Barrier in 1988, three more in 1989, and three more again in 1990, (Frantz, 1936 and Gerhold,,1992).

The Barrier feed and travel systems have overcome difficulties often experienced with the Isodynamic Separator, such as clogging of the feed mechanism and erratic movement of particles.

The feed system is mounted on the column of the base. The feed hopper is suspended over the receiving compartment of the chute. The feed and travel systems have separate

electrical circuits and controls. The chute is supported on a sturdy, movable carriage to which the chute vibrator is attached at an attitude which transmits vibration at an angle upward from the surface of the chute. The power supply, installed between the alternating current source and the separator/s coils, provides linked regulation of direct current and voltage. As resistance increases with heating of the coils, voltage increases automatically to maintain the selected current.

In normal intermittent operation stable currents up to about 1.8 Amperes can be maintained in ambient temperature up to approximately 32^{o} C (90^{o} F). With the former electrical system, without current regulation, maximum current at start-up of 1.8 A fell off to 1.5 A within one half-hour, (Frantz, 1936 and Gerhold,,1992).

Plate 2.12 Photograph of Standard Parts

Above photo features all standard parts for guiding particles into the Barrier field:

(a) Diamagnetic feed trough

(b) Steep side slope feed trough

(c) Feed blade

(d) Feed trough assembly

Plate 2.13

The hopper has an orifice one quarter inch in diameter above an adjustable platform. Material piling up on the platform stops flow out of the hopper until vibration is supplied, Plate 2.13.

The Frantz® Magnetic Barrier Laboratory Separator, Model LB-1

(Complete System) **Specifications, Plate 2.14**

(i) Electrical power required:

110-120V .a.c.,

3.0 A., 50-60 Hertz.

(ii) Size:

16 x 32 x 34" high

(41 x 81 x 86 cm

(iii) Approx. Net weight: 329 lbs (150 kg)

Plate 2.14. The Frantz® Magnetic Barrier Laboratory Separator, Model LB-1

2.7. Some Selected Industrial Flow Sheets:

(i) Flow Sheet of Wabush Mines in Canada:

Magnetic separators are commonly used for upgrading low-grade iron ores, wet high-intensity separation often replacing the flotation of hematite. A combination of magnetic and high-tension separation has been used at the Scully Mine of Wabush Mines in Canada (Anon., 1974), see Fig 2.14. The ore, is a quartz-specular hematite-magnetite schist, contains 35% Fe. After primary crushing and autogenous milling to sizes less than lmm, is fed to banks of rougher and cleaner spiral concentrators.The spiral concentrate is filtered and dried, and cleaned in high-tension roll separators. The spiral tailings are thickened, and further treated by magnetic drum separators to remove residual magnetite, followed by Jones wet high-intensity separators, which remove any remaining hematite. The magnetic concentrates are classified and dried, and blended with the high-tension product, to give a

final concentrate of about 66% Fe. Cleaning of gravity tailings products by magnetic separation is preferred, as relatively small amounts of magnetic concentrate have to be dealt with, the bulk of the material being unaffected by the magnetic field. Similarly, relatively little material is pinned to the rotor in the high-tension treatment of the gravity concentrate, the iron minerals being unaffected by the ionic field.

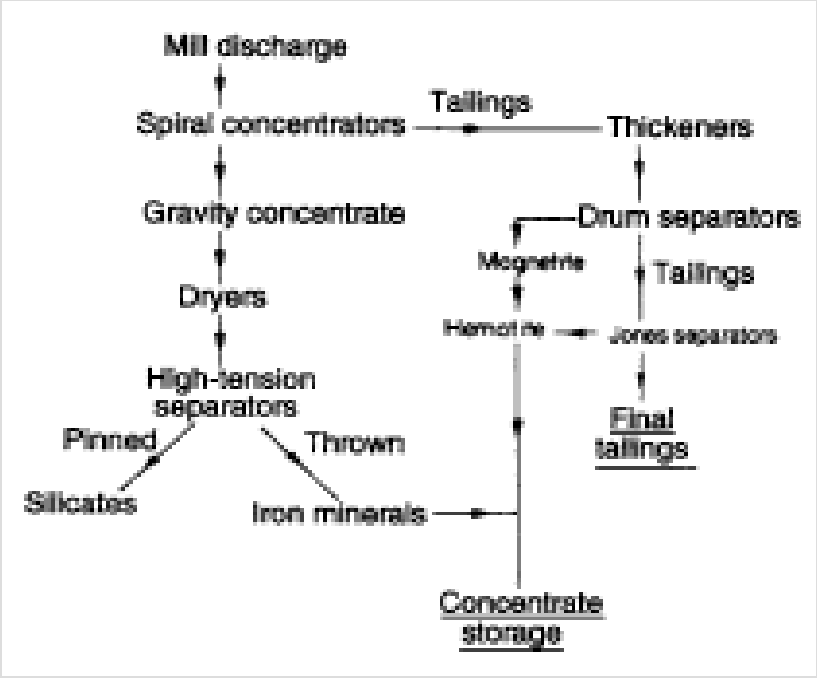


Fig. 2.14 Flowsheet of Scully concentrator

(ii) Flow Sheet of Hu Jia Miao Mines in China:

The Hu Jia Miao iron ore processing flow- sheet is shown in Figs 2.15 and 2.16. It could be seen in Tables 2.3 and 2.4, that the iron grains sizes are not even. When the ore is ground to -74 μm by53.95% yield, about 64.55% iron minerals and 57.89% gang minerals are liberated. The mineral processing flow sheet, shows that the ore is ground to -74 μm about 60% in a primary ball mill, then classified by cyclones to two fractions. The coarse fraction is treated with the spirals to take out most of the liberated iron minerals as final concentrate. The spirals tailings are scavenged with low intensity magnetic separator (LIMS) and 8 SLon-2000 MIMS (0.4T) which discharge 40.88% mass as final tailings containing only 9.57%Fe. The mags of SLon MIMS and the spiral middlings are ground further with a ball mill, then returned to the primary cyclone.

Table 2.3 Particle Crystal Size Distributions of Ru Jia Miao Iron Ore

Mineral	Crystal size/μm						
	2000-589	589-295	295-147	147-74	74-35	35-10	10-0
Iron minerals/(mass %)	2.14	8.26	14.69	22.63	23.27	23.08	5.93
Gang minerals/(mass %)	2.10	13.42	23.22	24.32	18.12	14.63	4.19

Table 2.4 Particle Size Liberation of Ru Jia Miao Iron Ore

Particles Size/ (Ground to -74 μm%)	**Liberation (Weight%)**	
	Iron Minerals	**Gangue Minerals**
53.59	64.55	57.89
73.32	84.88	68.01
80.37	87.56	74.56
90.54	94.04	85.77

The mags of the LIMS and SLon VP HGMS are cleaned further by reverse flotation. The collectors of the flotation are very sensitive to quartz and quartz- integrated minerals. Most of the quartz and un-liberated minerals can be removed from the iron concentrate reverse flotation. The operational costs of SLon magnetic separators are very low. For example, to treat one ton of feed iron ore, the SLon MIMS consumes only about 0.5 kWh electricity and 2 m^3 recycling water and the SLon VP HGMS consumes only about 1 kWh electricity and 2 m^3 recycling water. They can discharge about 68.63% mass of low-grade final tailings in the early stage. Spiral operational cost is also very low since 10.38% mass of spiral concentrate is considered as final iron concentrate. Flotation operation cost is a relatively high. But only about 20.99% mass containing 46.31%Fe goes into the flotation stage. SLon magnetic separators created very good conditions for flotation. They discharged most of the slimes and gangue minerals as final tailings, which otherwise would consume a lot of reagents and may damage the flotation process. They help flotation to get good results and greatly reduce reagents consumption. So flotation operation cost is relatively small for the total flowsheet. The LIMS mags and SLon mags of the fine fraction must be cleaned with reverse flotation instead of SLon magnetic separator. The reason is that in the feed of them there are a lot of magnetite and quartz associated particles. For example, if one particle contains 1% or more magnetite and 99% or less quartz, it will be captured by SLon magnetic separator as mags in the roughing stage and re-captured again

in the cleaning stage as mags, if SLon was applied as cleaner. It will enter into the final concentrate and decreased the final concentrate grade. But for reverse flotation, the collectors are very sensitive to quartz. For example, if a particle contains 10% quartz and 90% iron or other minerals, the collectors will capture it into the tailings. So reverse flotation can get very pure iron concentrate,(Young,2008). For the excellent performance of SLon magnetic separators and reverse flotation, the plant get very good results as: feed grade 23.10%Fe, concentrate grade 67.54%Fe, tailings assay 10.20% Fe, the iron recovery reaches 65.78%.

(iii) Flow Sheet for Patent No 3672579 in USA:

A process for upgrading low-grade magnetite-containing iron ore with minimum fine grinding has represented in aUS patent (No 3672579). The dry ore is first comminuted to between about three-fourths inch and 10 mesh particle size and magnetically separated. The dry tailing is discarded and the concentrate is ground to between about 20 and 100 mesh and again subjected to dry magnetic separation. The dry tailing is again discarded and the concentrate is then finely ground according to conventional practice and subjected to wet magnetic separation. About 90 percent or more of the tailing is discarded dry without fine grinding, permitting easy disposal without possible ecological damage due to lake and stream pollution, Fig 2.17, (Lawver and Minn. 1972).

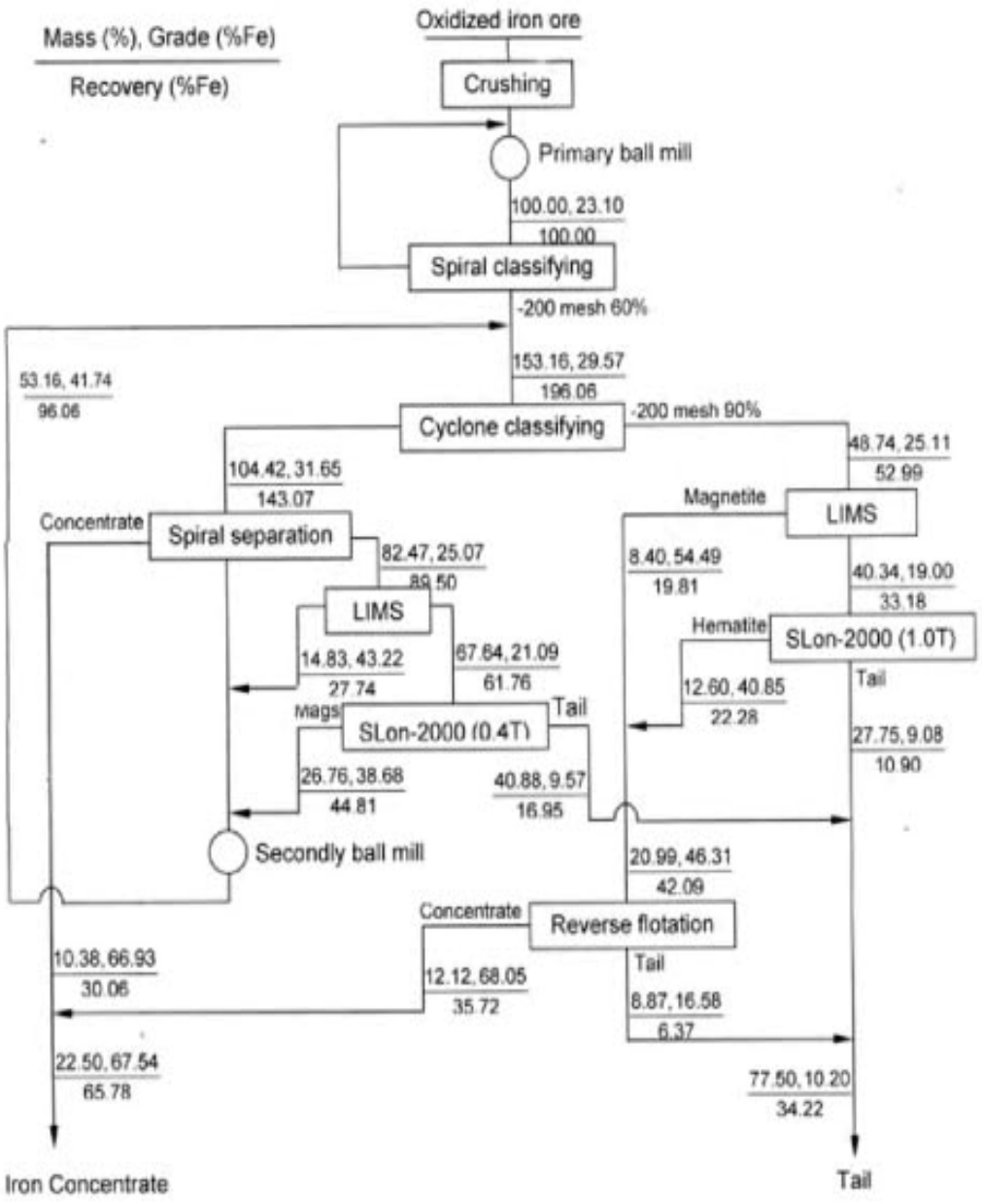


Fig. 2.15 The Hu Jia Miao Iron Ore Processing Flowsheet.

Fig. 2.16 The photo of SLon-2000 VP HGMS in Hu Jia Miao processing plant

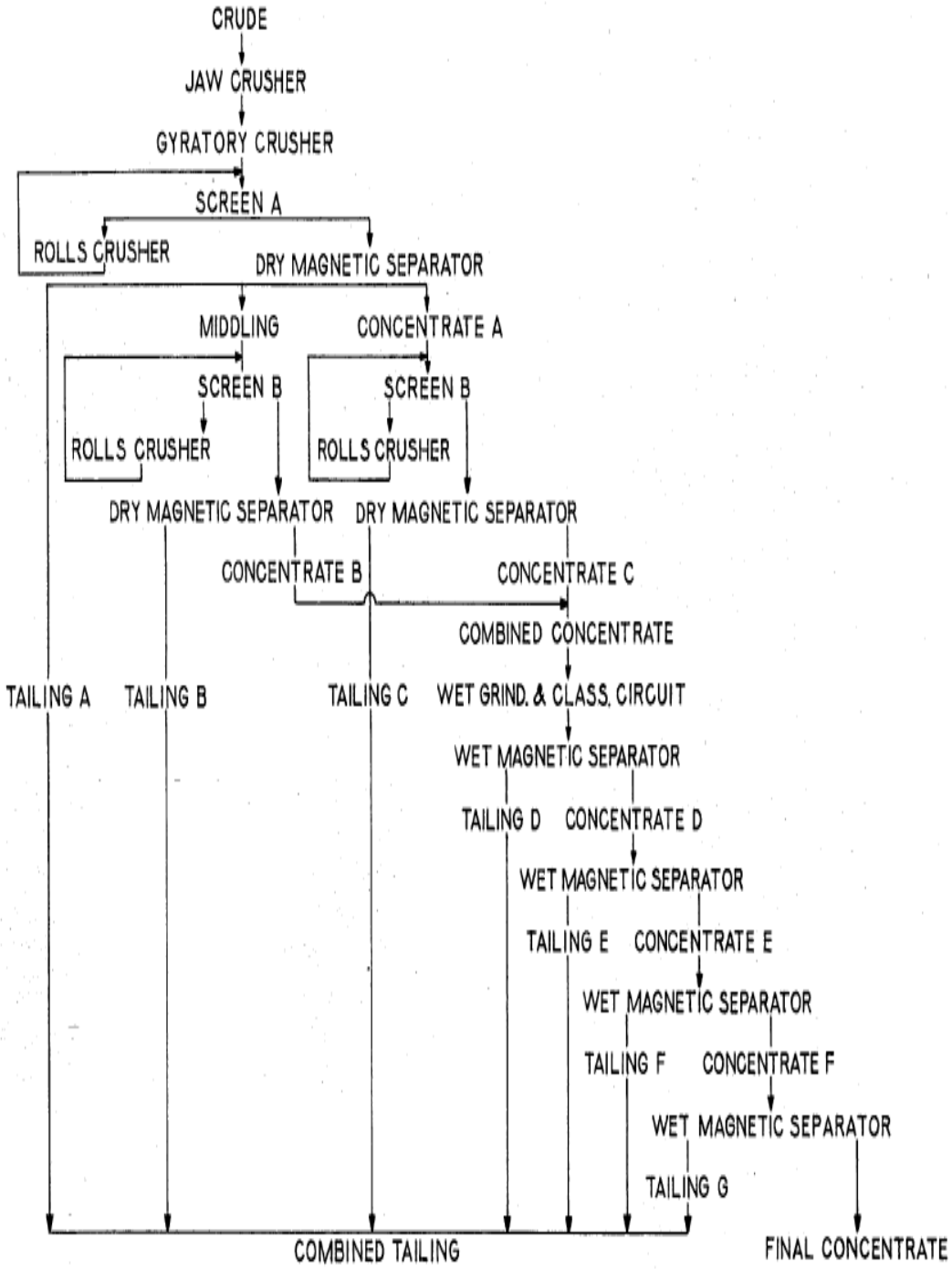

CRUDE
JAW CRUSHER
GYRATORY CRUSHER
SCREEN A
ROLLS CRUSHER
DRY MAGNETIC SEPARATOR
MIDDLING
CONCENTRATE A
SCREEN B
SCREEN B
ROLLS CRUSHER
ROLLS CRUSHER
DRY MAGNETIC SEPARATOR
DRY MAGNETIC SEPARATOR
CONCENTRATE B
CONCENTRATE C
COMBINED CONCENTRATE
TAILING A
TAILING B
TAILING C
WET GRIND. & CLASS. CIRCUIT
WET MAGNETIC SEPARATOR
TAILING D
CONCENTRATE D
WET MAGNETIC SEPARATOR
TAILING E
CONCENTRATE E
WET MAGNETIC SEPARATOR
TAILING F
CONCENTRATE F
WET MAGNETIC SEPARATOR
TAILING G
COMBINED TAILING
FINAL CONCENTRATE

Fig. 2.17

Chapter Three

Experimental Work

3.1. Introduction:

A laboratory research work was carried out within the University of Khartoum on the Bajrawiya iron ores in Sudan. The objective of this work was aimed at concentration of these ores which contain fine silica, sand and clay materials as main gangue materials in the Bajrawiya iron ores. Fig 3.1 shows a devised tentative flow-sheet for the laboratory work procedure.

3.2. Materials:

Iron ore samples were collected from the iron ore deposits of El Bajrawiya. The samples were taken from the different types of iron oxides occurrences; each sample weights about 20 kg. The first step in the experimental work was to determine the approximate mineralogical composition of these different ore oxides, at least in so far as the principal minerals are concerned. These was conducted by microscopic examination, AAS technique and Gravimetric method. The content of iron and the distribution of the different oxides were determined and given in Table 3.1.

Table 3.1 Chemical analysis of the received samples.

Constituents,%	Bog1&Bog2	Bog2-5	Ferricrete&Bog1-001	Oolitic
Fe	32.94	41.43	17.00	48.00
Fe_2O_3	47.09	59.23	24.30	68.63
MnO_2	0.115	2.326	0.216	0.221
CaO	0.442	0.827	0.282	0.309
SO_4	0.665	0.320	0.227	0.207
SiO_2&Others	51.69	37.29	74.97	30.63

3.3. Apparatus:

The following apparatus were employed to prepare the feed for conducting the separation, experimental work and both feed and separation products analyses.

1. Jaw Crusher (Dodge).
2. Disk Mill Model DM200.
3. Jones Riffle Sampler
4. RS 100 Vibratory Disc Mill
5. Auto divider
6. Drying Oven
7. Digital Balance
8. Standard Weighing

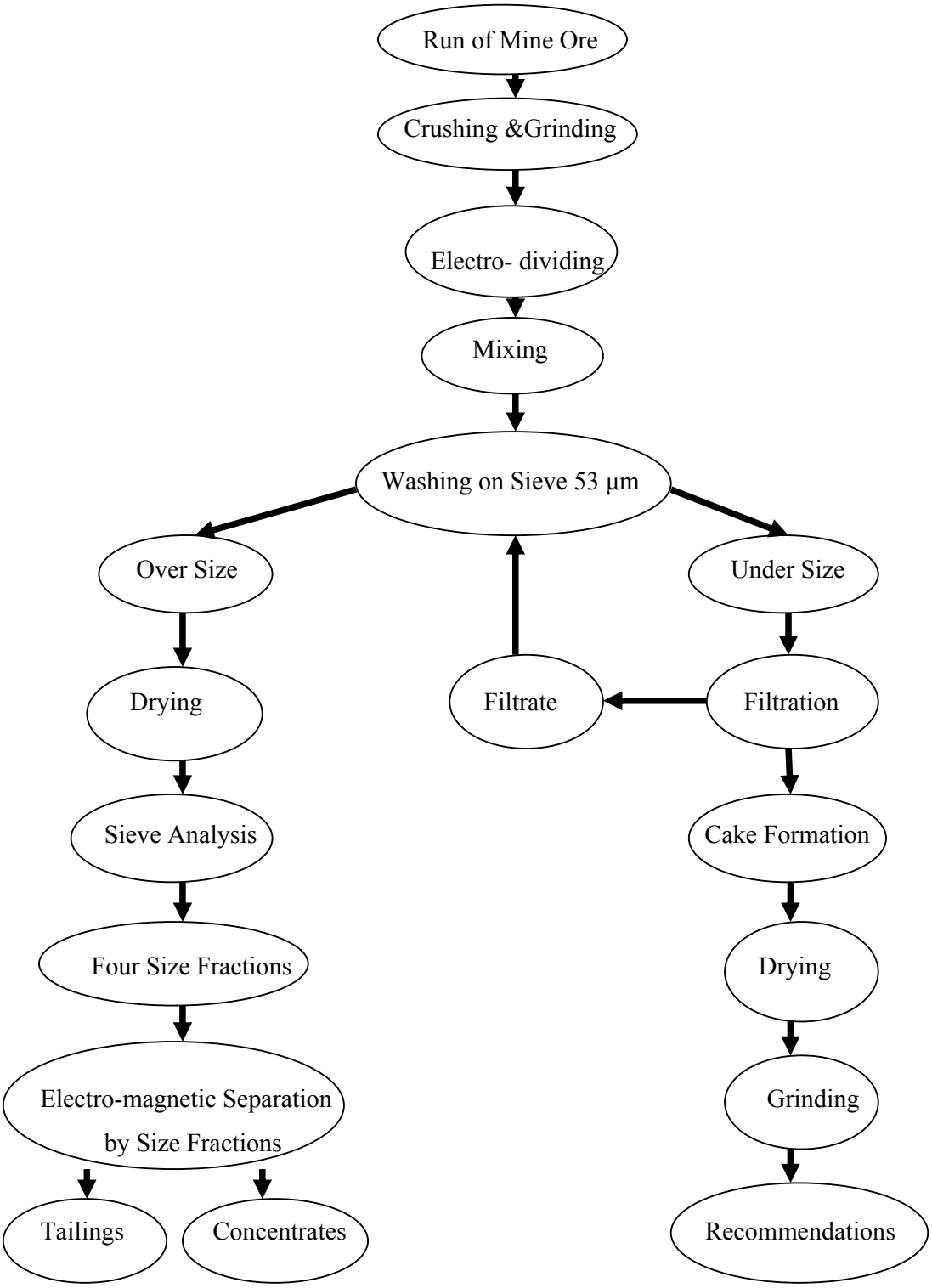


Fig 3.1 shows a devised tentative flow-sheet for the laboratory work procedure.

9. Atomic Absorption Spectrometer
10. Distilled Water Apparatus
11. pH Meter.
12. Standard Density Bottle.
13.Desicator
14. Digital tachometer,
15. Mixer Gives Various Speeds.
16. Stopwatch.
17. Stereomicroscope with Digital Camera.
18. Laboratory Test Filter.
19. Laboratory Ball Mill
20. Vibrating Testing-Sieve Shaker.
21. Magnetic Barrier Laboratory Separator, Model LB-1.

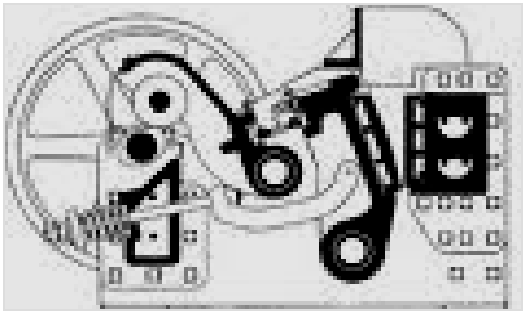


Fig. 3.2 Jaw Crusher

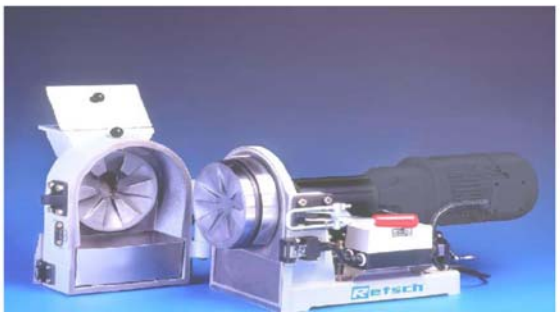


Fig. 3.3 Disk Mill Model DM200

Fig. 3.4 Jones riffle sampler

Fig. 3.5 RS 100 Vibratory Disc Mill

Fig. 3.6, a- Auto divider

Fig. 3.6, b- Auto divider

Fig. 3.7, a. Drying oven

Fig. 3.7, b. Drying oven

Fig. 3.8, a. Digital Balance Fig. 3.8, b Digital balance Fig. 3.9 Standard weighing

Fig. 3.10, Atomic Absorption Spectrometer

Fig. 3.11, Distilled water apparatus-
with beaker, glass funnel and bottle.

Fig 3.12, pH Meter

Fig. 3.13, Standard density bottles

Fig. 3.14, Desicator

Fig. 3.15. Digital tachometer,

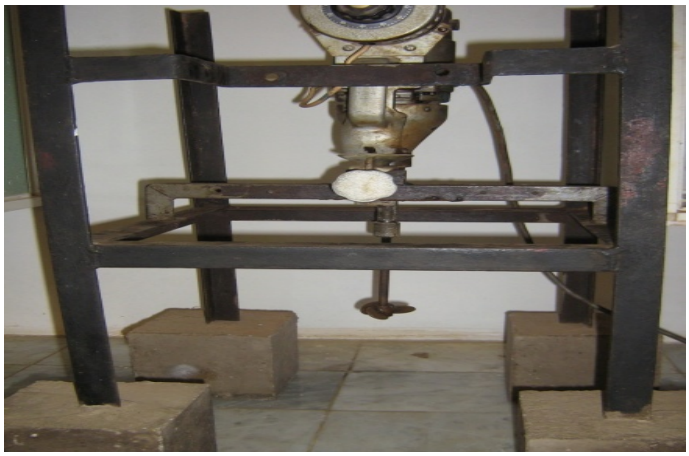


Fig. 3.16. Mixer with deferent speeds,

Fig. 3.17, Stopwatch,

Fig 3.18, Stereomicroscope with digital camera

Fig. 3.19, Laboratory test filter

Fig. 3.20, Laboratory ball mill

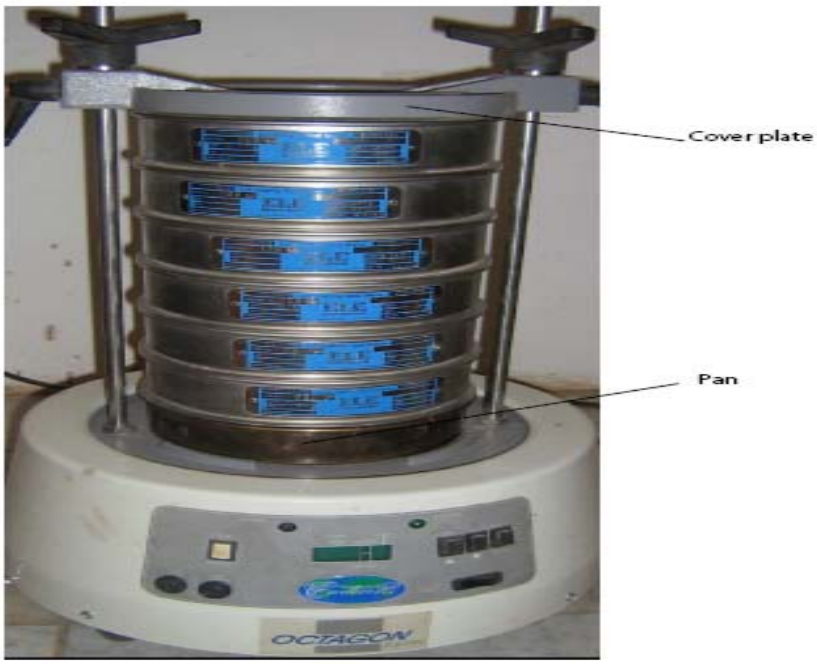


Fig. 3.21. Vibrating Testing-Sieve Shaker.

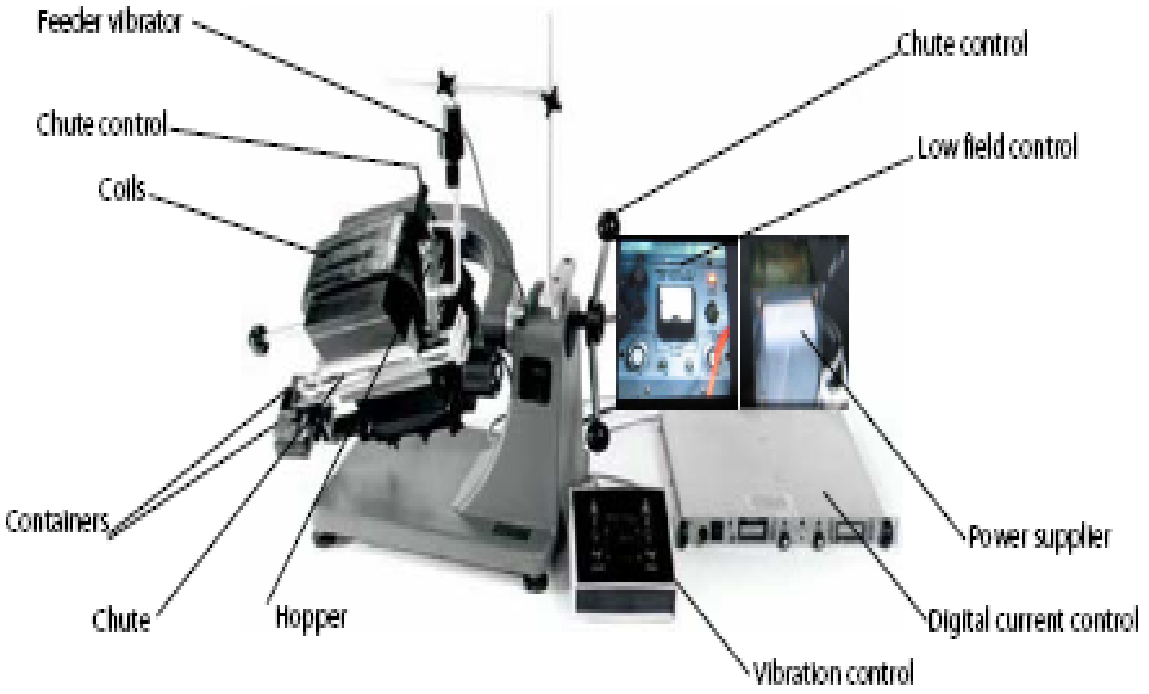


Fig. 3.22, Magnetic Barrier Laboratory Separator, Model LB-1.

3.4. Testing Procedures:

Investigation of the feed ore samples by physical and microcopies has shown that the feed samples contain considerable amount of clay materials and very fine iron ores, which would reduce the efficiency of the magnetic separation. Hence, it was decided to remove these fine materials from the feed prior to conduct the separation process. Removal of these materials was done by slurring, agitation and screening. These operations were carried out by the mixer which having various speeds and a 53 μm screen.

3.4.1. Samples Preparation:

The first mechanical stage of the processing of El Bajrawiya iron ores was the comminution, (crushing and grinding).

Both processes were performed as dry operations in two or three stages. The large lumps of the six types of the iron ores are shown in plate 3.1. These lumps of the feed ores were first disintegrated by hammer to produce suitable sizes for the jaw crusher. A dodge jaw crusher was used. On the other hand , Disk Mill Model DM200 was used for the reduction of the products of the jaw crusher.

Crushing was made in open circuit and 5 kg from each samples of El Bajrawiya iron ores was crushed in order to produce particles less than 2 cm, and then was divided using a large Jones riffle sampler. After, that all ores samples were fed into the grinding process, using RS 100 Vibarotary Disc Mill. The machine has been operated for five minutes in each stage of comminution. To increase the particles homogeneity after the comminution, auto divider has been used. The electrical centrifugal divider has the ability to precisely divide larger samples into working size samples in a matter of seconds and considered as accurated method of producing homogenous samples from dry granular or powdered material. The dividing and mixing are accomplished by the centrifugal action of a motor driven revolving disc which is positioned under the hopper.

Bog1

Bog2

Bog1-001

Bog2-5

Ferricrite

Oolitic

Plate. 3.1, El Bajrawiya iron ore samples

3.4.2. Determination of Moisture Content:

The samples were taken to the laboratory in order to crushing, grinding and dividing. After dividing of the samples was completed, Plate. 3.2, a 300 g from each sample type of the ores put in a vessel made from nickel, using the digital balance, with accuracy 0.1g , then these samples were dried in the drying oven, , and then weighed, the temperature of drying oven was adjusted at 105 C^o. The samples were weighed wet, then dried at a suitable temperature until all hygroscopic water is driven off, and then weighed again. The difference in weight represents moisture and is expressed as:

$$\%\ moisture = \frac{(\text{wet weight} - \text{dry weight})}{\text{wet weight}} * 100 \quad \ldots\ldots\ldots\

There are many methods used to measure the solid density, each method having its relative advantages and disadvantages. The method of standard density bottle is considered as an accurate method and cheaper than the other methods, therefore was used for determination of the solid density of the products.

3.4.4. Determination of Minerals Constituents in Feed and Products:

The samples are named Bog1, Bog1-001, Bog2, Bog2-5, Ferricrite, and Oolitic, The main gangue minerals are silica oxide and silicate minerals. The samples were assayed using Varian Spectra 220 Atomic Absorption Spectrometer and Gravimetric method. All chemical analyses for this research work were carried at the Geological Research Authority of Sudan, (GRAS), refer to the appendix A. For the elements and their standard atomic mass, refer to appendix B.

3.4.5. Mixing Process:

After grinding the samples were dried, cooled, and reweighed, then each sample was put in 500 ml bottle and the water was added until reached 500 ml, then the manual mixing rod was used, after that a few drops of water has been added to complete the volume of the bottle to 500 ml, then the mixture was poured in cylindrical vessel with cover and other 500 ml of water was added to the mixture usining plastic washing bottle. The vessel then has been put on a bench standing under the center of mixer. The time of mixing was15 min using mixer speed is around 2200 rpm.

3.4.6. Washing process:

The presence of contaminants in the iron ores such as manganese, phosphors, alumina, sulpher and silica have a negative effect on the steel production process, because the production costs become very high. The washing process can reduce the amounts of these contaminants in the feed material, then the processing of the iron ore may be viable as a result of the cost reduction by means of washing . In order to remove the most of fine impurities in the ground iron ores, these ores were mixed to wash these fines.

3.4.7. Filtration Process:

A vacuum filter paper test circuit is shown in Fig. 3.19. The filter paper, consisting of a section of the industrial filter medium, is connected to a filtrate receiver . The receiver is connected to a vacuum pump. The cycle is divided into three sections- cake formation (particles finer than 53 μm), drying, and discharge. While under vacuum, the test paper is

not submerged for the cake formation period in the pulp to be tested. The time of filtration was depended on the time of settling for the mixture. The paper was then removed and held in the crucible for the allowed drying time. Then the cake has been removed, weighed, and dried for each sample. The dried cakes also have been ground by laboratory ball mill, Fig. 3.20.

3.4.8. Dry Sieve Analysis:

The purpose of dry sieve analysis is to determine the grain size distribution for particles coarser than 53 μm and its assay by placing a weighed dry sample of the material to be tested on the top or coarse sieve of a nest. Shaking the nest until most of the undersize has passed the coarse sieve. The weights, retained on each sieve have been measured. The results are shown in Table 3.2

Table 3.2 shows the Size Analysis of the Feed Sample for Magnetic Separation

Ore type	*Sieve size range*		*Sieve fractions*		*Nominal aperture size*	*Cumulative %*	
	μm		*wt* (g)	*wt %*		*undersize*	*oversize*
Bog1 and Bog2	-1180 to	+250	10.53	3.52	250	96.47	3.525
	-250 to	+125	41.27	13.81	125	82.67	17.33
Dried mass	-125 to	+75	44.03	14.73	75	67.93	32.07
298.8333 g	-75 to	+53	25.17	8.42	53	59.51	40.49
	-53		177.60	59.43	Total(∑)	0.08	99.92
	Total(∑)		298.60	99.92			
Bog2-5	-853 to	+250	14.10	4.73	250	95.27	4.73
	-250 to	+125	41.10	13.80	125	81.47	18.53
Dried mass	-125 to	+75	33.30	11.18	75	70.28	29.71
297.8333g	-75 to	+53	23.00	7.72	53	62.56	37.44
	-53		186.13	62.50	Total(∑)	0.07	99.93
	Total(∑)		297.63	99.93			
Bog1-001	-600 to	+250	26.43	8.84	250	91.16	8.84
&Ferricrete	-250 to	+125	67.77	22.66	125	68.50	31.50
	-125 to	+75	40.50	13.54	75	54.95	45.05
Dried mass	-75 to	+53	24.60	8.23	53	46.72	53.28
299 g	-53		139.77	46.74	Total(∑)	-0.02	100.02
	Total(∑)		299.07	100.02			
Oolitic	-600 to	+250	17.13	5.74	250	94.26	5.74
	-250 to	+125	67.33	22.54	125	71.72	28.28
	-125 to	+75	42.90	14.36	75	57.35	42.65
Dried mass	-75 to	+53	25.63	8.58	53	48.77	51.23
298.6667g	-53		145.27	48.64	Total(∑)	0.13	99.87
	Total(∑)		298.27	99.87			

3.4.9. Electro-Magnetic Separation Process:

Electro-magnetic separation tests were done using a Magnetic Barrier Laboratory Separator Model LB-1. The samples were prepared to Group1 and Group2, each group consists of a mixture of particles coarser than 53 um, which have been included Bog1(50% by weight) and Bog2(50% by weight), Bog2-5 (100% by weight), Freecrite (50% by weight) and Bog1-001(50% by weight) and Oolitic (100% by weight). The data of Group1 is given in details in tables in chapter 4, but all the steps which have been made in Group1 were made for Group2. The method in which mineral particles are separated as they move along the length of the chute will depend on:

(i) The tilt of the chute.

(ii) The amperage applied, and

(iii) The slope and rate of feed to the chute

Group1 and Group2 were used for the following tests:

1) The test samples were crushed and powdered to minus 1,180 um and divided.

2) The ground samples were mixed with water and washed using screen 53 μm. Then they dried and screened through 250 μm, 125 μm, 75 μm and 53 μm.

3) The dry magnetic separation tests were made to each size fraction from Group1 and Group2 with 0.8 and 1.0 Amperes respectively.

4) Stereomicroscope with digital camera was used before and after each separation process in order to photo analysis.

3.4.9.1. The Equipment Used:

1) Magnetic Barrier Laboratory Separator, Model LB-1.

2) Digital balance with accuracy 0.1g

3) Stopwatch, .

4) Stereomicroscope with digital camera,

3.4.9.2. Procedure:

The prepared iron feed was fed into the hopper and piling up on the platform which was stopped flow out of the hopper until vibration is supplied. Wherever the feed hopper was suspended over the receiving compartment of the chute. The magnetic system was inclined so that gravity urges particles toward the far side of the chute by angle 12^{o} and down its length by angle 11. The power supply has been installed between the alternating current

source and the separator coils. Then the current of coils was adjusted to 0.8 Ampere for Group1 and 1.0 Ampere for Group2. After that feed and travel systems circuits were turned on, when the size fraction of each Group has been fed into the hopper. Therefore, size fraction of sample was undergone separation in the field of the Magnetic Barrier Laboratory Separator (Model LB-1).The light colored diamagnetic grains were deflected along the magnetic barrier and collected in white container, while the darker paramagnetic grains passed through it and out of the fields in the channel on the far side of the divider and received in dark container and reweighed. The time of each run was recorded; also the results of chemical analysis were recorded.

3.4.9.3. Assessment Method of the Results:

Mass variance and recovery variance have been computed according to equations were mentioned by Wills, 1984 and updated to an Excel Spreadsheets (MassVar and RecVar), where were programmed by JKTech Pty Ltd, 2005.

In order to assess testing performance, it is necessary to account for the products in terms of material and contained component weights. Mass balancing is particularly important in accounting for valuable mineral or metal distributions, and the two-product formula is of great use in this respect.

If the weights of the feed, concentrate, and tailings are F, C, and T respectively, and their corresponding assays f, c, and t, then

$$F = C + T \quad \text{---------- (3.3)}$$

i.e. material input = material output

and $$Ff = Cc + Tt \quad \text{---------- (3.4)}$$

i.e. the valuable metal (or mineral) is balanced.

Therefore, $Ff = Cc + (F - C)t$

which gives $$F/C = (c - t) / (f - t) \quad \text{---------- (3.5)}$$

Where F/C represents the ratio of concentration.

However, the recovery will be $(Cc/Ff) \times 100\%$

or $$= 100\, c(f - t) / f(c - t)\% \quad \text{---------- (3.6)}$$

As values of recovery, ratio of concentration and *enrichment ratio (c/f)* can be determined from the assay results alone, the two-product formulae are often used to provide information for testing performance.

3.4.9.4. Variance in Recovery:

Equation 3.6, defining recovery of a unit operation, is very sensitive to the value of t, as the equation represents the ratio of the two expressions c/f and (c- t)/(f-t), which differ only by the presence of t in the latter. Equation 3.7 can be partially differentiated with respect to f, c, and t to give:

$$\frac{\partial R}{\partial f} = \frac{100ct}{f^2(c-t)}$$
$$\frac{\partial R}{\partial c} = \frac{-100t(f-t)}{f(c-t)^2}$$
$$\frac{\partial R}{\partial t} = \frac{-100c(c-f)}{f(c-t)^2}$$

Since the variance of a function can be found from its derivatives:

$$V_{F(x)} = \sum_{i} \left(\frac{\partial F}{\partial x_i}\right)^2 V_{x_i} \qquad \text{.....(3.7)}$$

$$V_R = (\partial R/\partial f)^2\ V_f + (\partial R/\partial c)^2\ V_c + (\partial R/\partial t)^2\ V_t$$

where V_R, Vf, Vc, and Vt are the variances in R, f, c, and t respectively.

Therefore:

$$V_R = \frac{100^2}{f^2(c-t)^2}\left[\frac{c^2t^2}{f^2}\ V_f + \frac{(f-t)^2t^2}{(c-t)^2}V_c + \frac{c^2(c-f)^2}{(c-t)^2}V_t\right] \qquad \text{.....(3.8)}$$

3.4.9.5. Variance in Concentrate Product:

Equation 3.5 can be used to calculate the concentrate weight as a fraction or percentage (C) of the feed weight:

$C = 100(f - t)/(c - t)$ ..... (3.9)

Although expression 3.9 is very useful in material balancing, it is, like the recovery equation, prone to considerable error if the component values are not well separated. For example, a hydrocyclone is a separator which produces good separation in terms of contained water content, and of certain size fractions, but not necessarily in terms of

contained metal values. When all such data is available, the problem is often to decide which component would produce the most accurate material balance. If Equation 3.9 is partially differentiated with respect to f, c, and t respectively, then:

$$\partial C/\partial f = 100/(c-t)$$
$$\partial C/\partial c = -100(f-t)/(c-t)^2$$
$$\partial C/\partial t = -100(c-f)/(c-t)^2$$

From Equation 3.8, the variance in C, Vc can be determined from

$$V_C = (\partial C/\partial f)^2 V_f + (\partial C/\partial c)^2 V_c + (\partial C/\partial t)^2 V_t$$

$$= \left[\frac{100}{c-t}\right]^2 V_f + 100^2 \left[\frac{f-t}{(c-t)^2}\right]^2 V_c + 100^2 \left[\frac{(c-f)}{(c-t)^2}\right]^2 V_t \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots \text{(3.10)}$$

This equation is called the "Propagation of Variance" and is a useful general rule. As all of the key terms are differences, the measurements with the largest differences will usually provide the best definition. The best defined separation will usually be the mineral (or metal) of commercial interest balanced across the entire concentrator.

3.4.9.6. Maximizing the Accuracy of Two-Product Recovery Computations:

It has been shown that the recovery Equation (3.6) is very sensitive to the accuracy of the component values, and to the degree of separation that has taken place. Equation 3.6 can also be written as:

$R = Cc/f$ …… (3.11)

where

$C = 100(f-t)/(c-t)$ …… (3.12)

C represents the percentage of the total feed weight which reports to the concentrate. This value can often be calculated by using components other than the component whose recovery is being determined (Wills, 1985).

The choice of "mass-fraction" component can be determined by sensitivity analysis. Equation 3.12 can be written as:

$M = 100(a-d)/(b-d)$ …… (3.13)

where a, b, and d are the mass-fraction components in feed, concentrate, and tailings respectively. These components being independent of f, c, and t, and M is the value of C calculated from these components. Hence:

$R = Mc/f$ …… (3.14)

From Equation 3..10:

$$V_M = \frac{100^2}{(b-d)^2}\left[V_a + \left(\frac{a-d}{b-d}\right)^2 V_b + \left(\frac{b-a}{b-d}\right)^2 V_d\right] \quad \text{……(3.15)}$$

Providing that estimates of component variance are known, then V_M can be calculated. If a number of components (e.g. a complete size analysis) are available, Equation 3.15 can be used to select the least sensitive component as the mass-fraction component. The component will be that which produces the lowest value of relative standard deviation (RSD) in the mass calculations:

$$\mathrm{RSD}(M) = V_M^{1/2}/M \quad \text{…… (3.16)}$$

Having chosen the mass-fraction component, the value of required component recovery can be calculated from:

$R = 100c(a - d)/[f(b - d)]$…… (3.17)

The variance in calculation of recovery can be found from Equation 3.14, i.e.:

$$V_R = (\partial R/\partial M)^2 V_M + (\partial R/\partial c)^2 V_c + (\partial R/\partial f)^2 V_f$$

Therefore:

$$V_R = (c/f)^2 V_M + (M/f)^2 V_c + (Mc/f^2)^2 V_f \quad \text{…… (3.18)}$$

Providing that c and f are independent of b and a. Combining Equations 3.14 and 3.18:

$$V_R = \frac{100^2 c^2}{(b-d)^2 f^2}\left[V_a + \left(\frac{a-d}{b-d}\right)^2 V_b + \left(\frac{b-a}{b-d}\right)^2 V_d + \left(\frac{a-d}{c}\right)^2 V_c + \left(\frac{a-d}{f}\right)^2 V_f\right] \quad \text{……(3.19)}$$

the mass-fraction component corresponds to the recovery component, then Equation 3.8 should be used to express recovery variance.

3.5. Statistical Evaluation of the Results of El Bajrawiya Iron Ores:

3.5.1. Mean of the Total Assays of the Feed for the Particles Coarser than 53 um:

Each type of samples have three total assays of the feed for particles coarser than 53 um, the first total assay of the feed was computed from the analysis of the original samples and the computation of the analysis of the particles finer than 53 μm sieve resulted after washing and filtration processes, the second and third total assays of the feed were computed after the analyses of the tailings and the concentrates for the electromagnetic separation process. Also at each three readings, the low reading was omitted, the mean of the assay is given by:

$\bar{f} = (\sum f) / n$ ……(3.20)

where:

$\bar{f}$ Mean of the total assay of the feed for the particles coarser than 53 μm

f total assay of the feed for the particles coarser than 53 μm

n the number of the measurements

and the standard deviation of the total assay of the feed is given by:

$SDf = \{\{\sum (f - \bar{f})^2\} / n\}^{1/2}$ ……(3.21)

Also the relative standard deviation of the total assay of the feed is given by:

$RSDf = SDf / \bar{f}$ ……(3.22)

For the concentrate and tailings, the same equations could be used, however, instead of the weight and the assay of the feed, the weights and the assays of concentrate and tailings products being used.

3.5.2. Standard Deviation Balance Equation:

The standard deviation of the feed is known, the standard deviations of the concentrates and the tailings can be estimated by using the balance equation for the materials, also from this equation we can estimate the mean of the total assays of the concentrates and tailings for the particles coarser than 53 μm, and the second experimental values for the concentrates and tailings.

$SDf = (SDc * C + SDt * T) / F$ ……..(3.23)

Where:

$SDc = (c * SDf)/ f$ …….(3.24)

and:

$SDt = (F* SDf / T)- (SDc*C/ T)$ ……(3.25)

SDf standard devation of the total assays of the feed

SDc standard deviation of the total assays of the concentrates.

SDt standard deviation of the total assays of the tailings.

F, C and T Total weights of the feed, concentrates and tailings, respectively in the overall size fractions.

3.5.3. Mean of Total Assays of Concentrates and Tailings for the Particles Coarser than 53 µm:

The standard deviation equation is given by:

$SDc = \{\{ \sum (c - \bar{c})^2\}/ n\}^{1/2}$ …….....(3.26)

In this case n =1, therefore, we find that:

$\bar{c} = c + SDc$ …….(3.27)

and also:

$SDt = \{\{ \sum (t - \bar{t})^2\}/ n\}^{1/2}$ …….(3.28)

also, in this case n = 1, and therefore, we find that:

$\bar{t} = t + SDt$, ……. (3.29)

where:

$\bar{c}$ and $\bar{t}$ Mean of the total assays of the concentrates and tailing, respectively, for the particles coarser than 53 µm.

3.5.4. Relative Standard Deviation of the Total Assays of the Concentrates and the Tailings for the Particles Coarser than 53 µm:

Calculations are made therefore of the relative standard deviation (or RSD), also called coefficient of variation (CV) and most often expressed as a percentage:

$RSDc = (SDc /\bar{c})*100$ ……(3.30)

$RSDt = (SDt/ \bar{t})*100$ …….(3,31)

3.5.5. Confidence Level of the Mean:

Francis and Annick, 2007, were stated that, when the number of the measurements n is small, and if there are no systematic errors, the true mean μ can be quite different from the arithmetic mean $\bar{x}$. An estimation of the true mean must then be made by calculating a confidence level within which a probability is given (for example 95 per cent), that the real value x_0 will be included. The confidence level, around the mean $\bar{x}$, in order that the true mean μ is within (or x_0 in the absence of all systematic errors), is given by the following formula:

$$\bar{x} - \frac{t \cdot s}{\sqrt{n}} \leq \mu \leq \bar{x} + \frac{t \cdot s}{\sqrt{n}} \qquad \text{……..……..(3.32)}$$

Where :

$\bar{x}$ mean of the total assays of the feed or the concentrates or the tailings for the particles coarser than 53 μm

s standard deviation of the total assays of the feed or the concentrates or the tailings.

t statistical factor

In equation 3.32, Student's coefficient t is a statistical factor, which depends upon n and of the level of confidence chosen. If, as well as the mean $\bar{x}$, the true value x_0 (or the true mean μ) is also known, expression 3.32 will permit the calculation of the value of t, according to the degree of confidence chosen . A value of t larger than that indicated in Table 3.3, on the line corresponding to the value of n, will be due to a systematic error.

Each measurement x_i must be considered as the sum of the true value x_0 and an absolute experimental error value ε The absolute error ε_i of measurement i is thus expressed as:

$$\varepsilon_i = x_i - x_0 \qquad \text{……..……(3.33)}$$

If the true value x_0 is not known, which is usually the case in chemical analysis, then the experimental error of the measurement i being e_i, is calculated by replacing x_0 by the mean $\bar{x}$ in (3.33).

$$e_i = x_i - \overline{x} \qquad \text{……(3.34)}$$

e_i represents the algebraic difference between the mean and the ith measurement.

Table 3.3. List of t-values for various confidence intervals, Gauglitz and Dinh,(2003).

Degrees of Freedom (n-1)	*Confidence Interval*				
	50%	*80%*	*90%*	*95%*	*99%*
1	1.000	3.078	6.314	12.706	63.657
2	0.816	1.886	2.920	4.303	9.925
3	0.765	1.638	2.353	3.182	5.841
4	0.741	1.533	2.132	2.776	4.604
5	0.727	1.476	2.015	2.571	4.032

Chapter Four

Presentation and Discussion of the Results

4.1. Auto Divider Results:

Testes Conditions:

1. About 5 kg ground sample from each type of El Bajrawiya iron ores were used.
2. The slot opening of auto divider has been adjusted to 0.9cm.
3. The division circumference of the divider equals 23.5cm.
4. Manual agitation was made using steel rod to help the particles to flow under the slot opening.
5. The time of run ranges between 10 to 12 minutes.
6. The division ratio is given by

$$\frac{\text{slot opening}}{\text{division circumference} - \text{slot opening}} \quad \ldots\ldots(4.1)$$

$$= \frac{0.9}{23.5 - 0.9} = 1:26.11$$

This means that at a few seconds, one gram was distributed to 26.1 portions, which were divided into 6 containers, i.e. each container has 4.352 portions. Therefore, each sample was become more homogeneous, and ready to use.

4.2. Moisture Content, Washing, Filtration, Dry Sieve Analyses and Solid density Results:

Tests Conditions:

1. Temperature of drying oven was adjusted at 105^{o} C.
2. Time of drying of the ground materials was checked at 4 hours and between 4 to 6 hours for materials coarser than 53 µm sieve resulted from washing and more than 6 hours for materials which are finer than 53 µm .
3. Standing up time to measure dried samples checked at 30 minute.
4. Vibrator auto timer controlled at 30 minute.
5. Degree of vibration adjusted to 5 out of 8.
6. Tap water was added for each sample during the washing on sieve No 300 with diameter (19.1cm) was 4 to 4.5 liters, with pH (8.25) at temperature around 31^{o}C.
7. During the addition of water, manual agitation with brush used in order to accelerate the separation process; it took about 7 to 10 minute for each sample.

8. Filter paper No 312-211with diameter (10.15cm) was used to separate materials finer than 53µm resulted after washing from solids as fine as 10 um . The process took between 10 to 30 minute, depended on the standing up time for each sample

9. Standard density bottle was used,(bottle No 455 with volume 50cm^3)

10. The temperature and the pH of double distilled water, which have been used in the tests, were 30.7^o C and 6.5 respectively.

4.2.1. Results of Moisture Contents:

Table 4.1 below shows the results of moisture contents after the ground samples have been divided,

Ore Type	Sample Weight(W), g	Dried Mass(D), g	Weight of Loss (W - D), g	% Moisture Content (100 x (W - D) / D)
Bog1 and Bog2	300	298.83	1.2	0.4
Bog2-5	300	297.80	2.2	0.7
Ferricrete and Bog1001	300	299	1.0	0.3
Oolitic	300	298.70	1.3	0.4

4.2.2. Results of Wet Screening:

After the samples were ground to liberate the iron ores oxides particles from the grains of the gangue materials, then the ground samples were divided and dried. The dried materials were wet-screened to know the size distribution of the ground materials.

Prior to grinding all samples for the dry magnetic separation testing, it was suggested in the different published researches and according to the practice, works that dry magnetic processes would produce reasonable results, when the feed sizes are less than 1cm and greater than 53 µm. This has made us to take a decision; only materials of sizes greater than 53 µm would be used for the concentration tests.

Using 53-µm screen, the wet-screening process gave the results presented in Table 4.2, for the several ore types.

These materials obtained from the screening process were dried in the drying Oven at 105^o C.

4.2.3. Results of Size Distribution (Group1):

The portions of the feed which having sizes over 53 um were dry screened to determine the size distribution of these feeds resulted from the grinding process. Tables from 4.2 to 4.6 show these results for all iron ore types.

It is worth to note that these materials (-1,180 +53, -850+53 and -600+53 µm) were dried, using the drying oven.

Microscopic and visual inspection of the products of the (-1,180 +53, -850+53 and -600+53 µm) and screening have shown that each particle size differs from the others, therefore it was suggested to test each size separate from the other sizes in the magnetic apparatus.

This conclusion was confirmed by testing all material sizes (-1,180 +53, -850+53 and -600+53 µm) together. The metallurgical results. i.e., recovery, grade and yield were unsatisfactorily. Hence separation each size alone in the magnetic apparatus was enhanced.

Table 4.2, shows the results of wet screening on a 53 µm screen.

Sample Name	Retained Weight (-1,180 +53, -850+53 and-600+53 µm), gram	Passed Weight (-53 um) gram
Bog1 and Bog2 (50%-50%)	123.43	175.4
Bog2-5	114.97	182.86
Ferricrete and Bog1-001	165.33	133.67
Oollitic Ore	158.70	139.97

Table 4.3 size distribution of Bog1 and Bog2 (feed each 50%) resulted from wet screening (over 53 µm).

Sieves Openings, (µm)	Average Retained Weights (gram) of Three Tests from Each Samples.
250	10.53
200	10.14
125	30.87
100	18.73
75	25.30
53	25.17
-53	2.20
Total	123.20

Table 4.4 size distribution Bog2-5 for sizes over 53 um resulted from wet screening.

Sieves Openings, (μm)	Average Retained Weights (gram) of Three Tests from Each Samples.
250	14.1
200	12.97
125	28.13
100	13.10
75	20.20
53	23.00
-53	3.33
Total	114.83

Table 4.5 size distribution of Ferricrete and Bog1-001 (feed each 50%) resulted from wet screening (over 53 um).

Sieves Openings, (μm)	Average Retained Weights (gram) of Three Tests from Each Samples.
250	26.43
200	21.33
125	46.43
100	17.67
75	22.83
53	24.60
-53	6.10
Total	165.37

Table 4.6 size distribution of Oolitic Ore for sizes over 53 um resulted from wet screening.

Sieves Openings, (μm)	Average Retained Weights (gram) of Three Tests from Each Samples.
250	17.13
200	22.20
125	45.13
100	18.93
75	23.97
53	25.63
-53	5.27
Total	158.27

4.2.4. Results of Size Distribution (Group2):

Table 4.7 below shows the weight retained for three tested samples, which were prepared from each sample of Group2.

Table 4.7

Sieves openings, (μm)	Weight retained (g)			
	Bog1 and Bog2	Bog2-5	Ferricrete&Bog1-001	Oolitic
250	30.80	40.90	83.70	52.80
125	127.40	122.40	210.30	221.60
75	141.80	112.70	123.80	126.10
53	67.00	58.70	67.20	63.00
Sum	367.00	334.70	4850	463.50

4.2.5. Results of Density Determination:

The solid density of each ground sample was determined. And also, a series of tests were carried out on the different sizes resulted from screening process and magnetic separation process in order to determine the density of each size fraction.

Table 4.8 below demonstrates the solid densities before washing, which were measured for El Bajrawiya samples with accuracy 0.01g.

Table 4.8

Ore type	Solid density (before wash) (g/cm^3)
Bog1 and Bog2	3.20
Bog2-5	3.54
Bog1-001 and Ferricrete	2.96
Oolitic	3.57

Tables 4.9 to 4.12 below demonstrate the solid densities after washing process, which were measured for El Bajrawiya samples for the various size fractions with accuracy 0.01g.

Table 4.9 shows the densities of the size fractions of Bog1 and Bog2 ores mixture.

Size fraction, (μm)	Solid density of materials, (g/cm3)
-1,180 +250	3.41
-250 +125	3.28
-125 +75	3.15
-75 +53	3.17
-53 +10	3.00

Table 4.10 shows the densities of the size fractions of Bog2-5 ore.

Size fraction, (μm)	Solid density of materials, (g/cm3)
-853 +250	3.58
-250 +125	3.61
-125 +75	3.41
-75 +53	3.54
-53 +10	2.90

Table 4.11 shows the densities of the size fractions of Ferricrete and Bog1-001 ores mixture.

Size fraction, (μm)	Solid density of materials, (g/cm3)
-600 +250	2.96
-250 +125	2.92
-125 +75	2.98
-75 +53	2.83
-53 +10	2.55

Table 4.12 shows the densities of the size fractions of Oolitic ore.

Size fraction, (μm)	Solid density of materials, (g/cm3)
-600 +250	3.62
-250 +125	3.65
-125 +75	3.86
-75 +53	3.52
-53 +10	3.03

Tables 4.13 to 4.16 below show the solid densities which were computed from the concentrates for four size fractions of each sample of El Bajrawiya iron ores(Group1 and Group2) after the magnetic separation process.

Table 4.13 shows the concentrated densities of the size fractions of Bog1 and Bog2 ores mixture.

Size fraction, (μm)	Solid density of materials, (g/cm3)	
	Group1	Group2
-1,180 +250	3.53	3.50
-250 +125	3.55	3.53
-125 +75	3.55	3.51
-75 +53	3.54	3.53

Table 4.14 shows the concentrated densities of the size fractions of Bog2-5 ore.

Size fraction, (μm)	Solid density of materials, (g/cm3)	
	Group1	Group2
-853 +250	3.85	3.67
-250 +125	3.90	3.84
-125 +75	3.90	3.88
-75 +53	3.89	3.92

Table 4.15 shows the concentrated densities of the size fractions of Ferricrete and Bog1-001 ores mixture.

Size fraction, (μm)	Solid density of materials, (g/cm3)	
	Group1	Group2
-600 +250	3.30	3.28
-250 +125	3.62	3.54
-125 +75	3.78	3.84
-75 +53	3.84	3.75

Table 4.16 shows the concentrated densities of the size fractions of Oolitic ore.

Size fraction, (μm)	Solid density of materials, (g/cm3)	
	Group1	Group2
-600 +250	3.86	3.87
-250 +125	3.86	3.92
-125 +75	4.03	3.90
-75 +53	3.89	3.91

4.2.6. Results of Chemical Analyses of Washing Process

As it is mentioned early that the microscopic investigation of all samples from the different ore types has indicated the fine sands and clay materials bond with the iron oxides grains. The next step of experimental work thereby to wash the feed in order to disintegrate its particles and hence the fine sands and clay would be removed during the washing process which is followed by screening process.

Washing was conducted in the high speed mixer which having 2200 rpm. About 300 grams of the ground iron ore were slurred with tap water to make litre of slurry for the ore and water. After slurring the feed, the mixer turned off and the slurry was screened on a 53 µm screen. The results of the chemical analyses are presented in Tables 4.17 and 4.18. However the analyses of the feed before washing were given in chapter 3.

Table 4.17 shows the chemical analyses of the feed after washing for the size fractions (-1,180 +53, -850+53 and -600+53 µm).

Constituents, %	Bog1and Bog2	Bog2-5	Ferricrete and Bog1-001	Oolitic	Feed type
Fe	32.33	45.57	14.56	50.79	Group1
Fe	32.03	44.71	12.95	50.88	Group2

Table 4.18 shows the chemical analyses of the feed after washing for the size fraction (-53+10 µm).

Constituents,%	Bog1&Bog2 (S1.5)	Bog2-5 (S2.5)	Ferricrete&Bog1-001 (S3.5)	Oolitic (S4.5)
Fe	33.08	42.65	21.65	45.44
Fe_2O_3	47.29	60.98	30.95	64.97
MnO_2	0.13	1.53	0.22	0.23
CaO	0.35	0.70	0.35	0.27
SO_4	0.11	0.25	0.09	0.28

The computations of the chemical analyses of the particles coarser than 53 µm sieve resulted after washing process and the chemical analyses of the particles finer than 53 µm sieve resulted after washing process are shown in Tables 4.19 and 4.20, respectively.

Table 4.19 shows computation of the chemical analyses of the feed after washing for the size fractions (-1,180 +53, -850+53 and -600+53 µm).

Constituents,%	Bog1&Bog2	Bog2-5	Ferricrete&Bog1-001	Oolitic
Fe	33.37	40.85	13.20	50.78
Fe_2O_3	47.71	58.40	18.87	72.61
MnO_2	0.10	3.71	0.21	0.21
CaO	0.584071	1.06	0.23	0.34
SO_4	1.47591	0.45	0.35	0.14

Table 4.20 shows computation of the chemical analyses of the feed after washing for the size fraction (-53 μm).

Constituents,%	Bog1&Bog2	Bog2-5	Ferricrete&Bog1-001	Oolitic
Fe	32.64	41.78	21.33	45.07
Fe_2O_3	46.66	59.73	30.50	64.44
MnO_2	0.13	1.49	0.22	0.23
CaO	0.34	0.68	0.34	0.27
SO_4	0.11	0.24	0.09	0.28

4.3. Statistical Parameters Results:

Table 4.21 The standard deviation, the mean and the relative standard deviation of the total assay of the feed for the particles coarser than 53 μm

Constituents,% Fe	Bog1and Bog2	Bog2-5	Ferricrete and Bog1-001	Oolitic	Feed type
f1	33.37	omitted	13.20	omitted	computed
f2	32.33	45.57	14.56	50.79	Group1
f3	Omitted	44.71	omitted	50.88	Group2
$\bar{f}$	32.85	45.14	13.88	50.84	
SDf	0.52	0.43	0.68	0.05	
RSDf	1.58	0.95	4.90	0.09	

Also, by using the equations from 3.23 to 3.31 (chapter 3), we can obtain upon the figures in table 4.22 below.

Table 4.22 The standard deviation, the mean and the relative standard deviation of the total assays of the concentrates and tailings for the particles coarser than 53 μm.

Constituents,% Fe	SDc	SDt	$\bar{c}$	$\bar{t}$	RSDc	RSDt
Bog1and Bog2	0.64	0.23	40.43	14.68	1.58	1.58
Bog2-5	0.47	0.27	50.29	28.72	0.93	0.93
Ferricrete and Bog1-001	2.23	0.21	49.90	4.71	4.46	4.46
Ooliti	0.05	0.04	52.81	42.75	0.10	0.10

Where :

f1 Assay of the particles coarser than 53 um sieve resulted after washing process.

f2 Assay of the particles coarser than 53 um sieve resulted after electromagnetic separation process for Group 1.

f3 Assay of the particles coarser than 53 um sieve resulted after electromagnetic separation process for Group 2.

$\bar{f}$, $\bar{c}$ and $\bar{t}$ Mean of assays for the feed, concentrates and tailings, respectively.

SDf, SDc and SDt Standard deviation for assays of feed, concentrates and tailings, respectively.

RSDf, RSDc and RSDt Relative standard deviation for assays of feed, concentrates and tailings, respectively.

4.4. Presentation of Magnetic Separation Results:

After preparation of the feed from the different ore types and producing various size fractions from each type, (see plates, 4.1, 4.2, 4.3 and 4.4). The effects of the sizes of particles on the performance of magnetic separation process could be then qualified accordingly. Each type of ore feed has yielded four fractions. Therefore each fraction tested separately under two groups of conditions by the magnetic separator. Two parameters which are considered to be utmost importance were: (i) Ampere which is reflection of the field intensity and (ii) flow rate of the feed introduced to the separator. While the seven other parameters were kept constant during the testing works.

Table 4.23 below gives the parameters under which were kept constant during conducting processing tests.

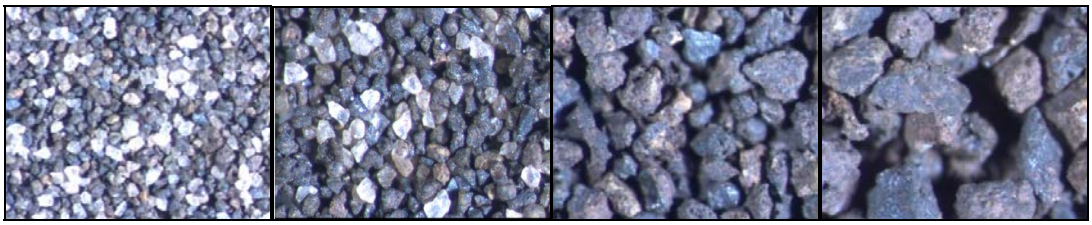
No	Parameters	Values
1	Tilting of chute(Degree)	12
2	Vibration of chute	8 out of 9
3	Slope of chute(Degree)	11
4	Width of chute(cm)	1.9
5	Vibration of hopper	4 out of 9
6	Lower internal diameter of hopper(cm)	0.5
7	Upper internal diameter of hopper(cm)	4.4

Table 4.24 below shows lists of the varied parameters.

No	Tests Group	Group1	Group2
	Varied Parameters	Values	Values
1	Current (Ampere)	0.8	1.0
	Voltage(volt)	54.4	70.5
2	Flow rate (g / minute)		
	Bog1 and Bog2	19.63	16.82
	Bog2-5	20.32	20.54
	Ferricrete and Bog1-001	25.66	30.09
	Oolitic	12.37	32.11

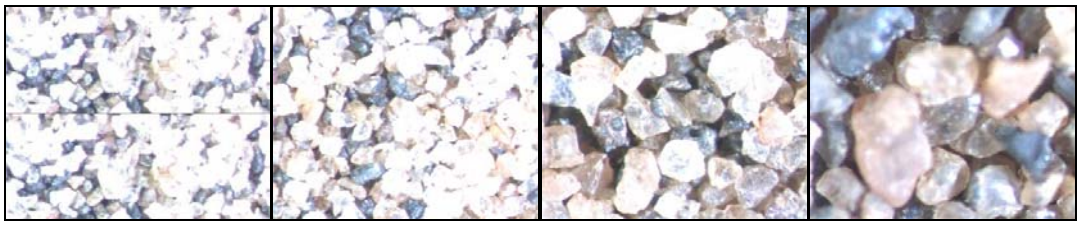
S1.1 S1.2 S1.3 S1.4

Plate 4.1 shows four size fractions of Bog1 and Bog2



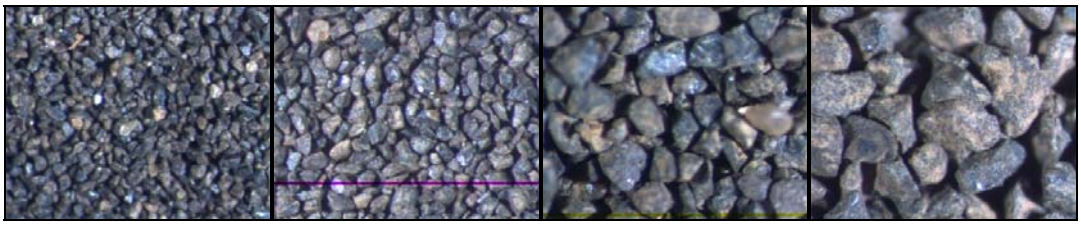
S2.1 S2.2 S2.3 S 2.4

Plate 4.2 shows four size fractions of Bog2-5



S3.1 S3.2 S3.3 S3.4

Plate 4.3 shows four size fractions for Ferricrete and Bog1-001



S4.1 S4.2 S4.3 S4.4

Plate 4.4 shows four size fractions for Oolitic

4.4.1. Results of Group (1):

The very important unwanted contaminant within raw iron ore is silica. The problem here relates to the fact that silica requires extremely high temperatures in order for it to be melted. Therefore, the more silica that exists in the raw material, the more energy is used in producing iron metal. For this purpose, electro-magnetic separation is selected to carry out the work of the separation in order to beneficiate the Bajrawiya iron ores. The obtained results from the magnetic separation are given in Tables 4.25 to 4.36

4.4.1.1. Bog1 and Bog2:

Table 4.25 shows the results of Bog1 and Bog2 from the magnetic separation (Group1).

Size Fractions(n)	Feed		Concentrate		Tailings	
(µm)	Yield (%)	Assay, Fe %(S1.n)	Yield(%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-1180 to+250	8.71	31.21	75.63	37.97	24.37	10.24
-250 to +125	34.10	36.32	80.53	41.34	19.47	15.56
-125 to +75	36.39	29.59	69.42	37.85	30.58	10.85
-75 to +53	20.80	31.52	54.04	42.51	45.96	18.60
Average		32.33	70.55	39.79	29.45	14.45

Table 4.26 shows the contaminants results of Bog1 and Bog2 from magnetic separation (Group1).

Constituents,%	CS1.1	CS1.2	CS1.3	CS1.4
MnO_2	0.0804	0.0724	0.0885	0.0664
CaO	0.012033	0.011194	0.009095	0.006716
SO4	0.13	0.14	0.11	0.05

Where:-

CS1.n Concentrates of sample (1), for size fraction (n).

TS1.n Tailings of sample (1), for size fraction (n).

S1.n Feed of sample (1), for size fraction (n)

Where, the range of size fractions are -1,180 to +250, -250 to +125, -125 to +75, and -75 to +53 µm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratios = F/C was calculated and found to be 1.42.

Where:-

F The yield of feed (%), C The yield of concentrate (%).

4.4.1.1. 1. Analysis of the Statistical Parameters Results:

The average values of the assays of the feed, concentrates and tailings are 32.85 %, 40.43 % and 14.68 %, respectively, and the standard deviation of the total assays of the feed, concentrates and tailings are 0.52 %, 0.64 % and 0.23 %. With 95 % level of confidence we find, the value of t = 2.571. We can then calculate:

$(t * SDf)/ n^{1/2} = 0.55$ %, and

$(t * SDc)/ n^{1/2} = 0.67$ %, and

$(t * SDt)/ n^{1/2} = 0.24$ %

These results determine a range of 32.85 ± 0.55 %, 40.43 ± 0.67 % and 14.68 ± 0.24 %, for the feed, concentrates and tailings respectively, when we fixed a level of confidence of 95 %, in which we have a 95% chance of finding the true average. There is probably a systematic error in these experiments. The value of 40.43 + 0.67 % would be included in this interval and would thus be considered as not a viable result.

4.4.1.1.2. An Excel Spreadsheets Results, (MassVar and RecVar):

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

RecVar calculates the error associated with the two-product recovery formula for the ***assay recovery***

Assay:		
Feed	32.85	%
Conc	40.43	%
Tail	14.68	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	1.58	%	0.52
Conc	1.58	%	0.64
Tail	1.58	%	0.23

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	0.01
B	326.43
C	107.30
D	141.64

Recovery:	86.85	%
Variance:	1.95	
SDs:	1.40	%

Calculated assay recovery
Variance of the assay recovery
Standard deviation of the assay recovery

MassVar : Estimation of errors in two-product mass flowrate

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

MassVar calculates the error associated with the two-product recovery formula for the ***mass recovery***

Assay:		
Feed	32.85	%
Conc	40.43	%
Tail	14.68	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	1.58	%	0.52
Conc	1.58	%	0.64
Tail	1.58	%	0.23

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	3.88
B	2.74
C	
D	1.14

Yield:	70.56	%	Calculated mass recovery
Variance:	7.20		Variance of the recovery
SDs:	2.68	%	Standard deviation of the recovery

4.4.1.1.3. Analysis of the Excel Spreadsheets Results:

Equation 3.8 is useful in assessing the error that can be expected in the calculated value of recovery due to errors in the measurement of feed, concentrates, and tailings . The Magnetic Barrier Laboratory Separator, Model LB-1, has been used with current equals to 0.8 Amper and average flow rate for the thin fractions is equal to19.63 g / minute, to treat a feed containing 32.85 % Fe to produce a concentrate grading 40.43 % Fe and a tailing of 14.68 % Fe , the calculated value of recovery (Equation 3.6) is 86.85 , and by comparing an equation 3.8 , chapter 3, by an Excel Spreadsheets (Recovery Variance), we can obtain upon:

$$V_R = 0.01\,(326.43V_f + 107.30\,V_c + 141.64V_t) \quad \text{.....(4.2)}$$

It is immediately apparent that the calculated value of recovery is most sensitive to the variance of the feed assay and the variance of the tailings than the variance of the concentrate assay. When, the feed, concentrate, and tailings have equal relative standard deviation of 1.58 %, then the standard deviations of feed, concentrate, and tailings are 0.52,

0.64, and 0.23 % respectively, and, from Equation 4.2., V_R = 1.95 , or the standard deviation of R is 1.40. This means that, to within 55.98 % confidence limits, the recovery is 86.85 ±1.40, and the concentrate is 40.43±0.64. Therefore, The value of C determined from iron ore assays (Equation 3.9, Chapter 3) is 70.56 %, and, from (Equation 3.10 chapter 3), V c = 7.20. The standard deviation, s, is thus 2.68 %, and the relative standard deviation in the mass calculation *(s/C)* is 0.038, with 93.903 % confidence level. also, we found that ,the concentrates of good quality can be obtained by grinding the ore within the size fraction of -250 to +125, as assays of the tailings, such as Fe is lower than assays of the particles finer than this size, also impurities such as MnO_2, and SO_4 were found to go into the tailings, as resulted of the washing process, which has been predated this process.

4.4.1.2. Bog2-5:

Table 4.27 shows the results of Bog2-5 from the magnetic separation (Group1).

Size Fractions(n)	Feed		Concentrate		Tailings	
(μm)	Yield (%)	Assay, Fe %(S1.n)	Yield(%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-853 to +250	12.65	45.77	89.13	46.20	10.87	42.29
-250 to +125	36.86	48.82	96.84	49.50	3.16	28.08
-125 to +75	29.87	46.07	81.78	53.53	18.22	12.59
-75 to +53	20.63	39.41	42.46	50.97	57.54	30.87
Average		45.57	80.15	49.82	19.85	28.45

Table 4.28 shows the contaminants results of Bog2-5 from magnetic separation (Group1).

Constituents,%	CS2.1	CS2.2	CS2.3	CS2.4
MnO_2	0.7274	0.7144	0.3446	0.655
CaO	0.122431	0.179378	0.096125	0.072619
SO4	0.134	0.102	0.172	0.198

Where:-

CS2.n Concentrates of sample (2), for size fraction (n).

TS2.n Tailings of sample (1), for size fraction (n).

S2.n Feed of sample (2), for size fraction (n).

Where, the range of size fractions are -853 to +250, -250 to +125, -125 to +75, and -75 to +53 μm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratio = F/C was calculated and found to be 1.25.

Where:-

F The yield of feed (%), C The yield of concentrate (%).

4.4.1.2.1. Analysis of the Statistical Parameters Results:

The average values of the assays of the feed, concentrates and tailings are 45.14 %, 50.29 % and 28.72 %, respectively, and the standard deviation of the total assays of the feed, concentrates and tailings are 0.43 %, 0.47 % and 0.27 %. With 95 % level of confidence we find, the value of t = 2.571. We can then calculate:

$(t * SDf)/ n^{1/2} = 0.45$ %, and

$(t * SDc)/ n^{1/2} = 0.49$ %, and

$(t * SDt)/ n^{1/2} = 0.28$ %

These results determine a range of 45.14 ± 0.45 %, 50.29 ± 0.49 % and 28.72 ± 0.28 %, for the feed, concentrates and tailings respectively, when we fixed a level of confidence of 95 %, in which we have a 95% chance of finding the true average. There is probably a systematic error in these experiments.

The value of 50.29 + 0.49 % would be included in this interval and would thus be considered as a viable result.

4.4.1.2.2. An Excel Spreadsheets Results, (MassVar and RecVar):

RecVar : Estimation of errors in recovery calculations

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

RecVar calculates the error associated with the two-product recovery formula for the ***assay recovery***

Assay:		
Feed	45.14	%
Conc	50.29	%
Tail	28.72	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	0.95	%	0.43
Conc	0.93	%	0.47
Tail	0.93	%	0.27

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	0.01
B	1023.79
C	477.99
D	144.17

Recovery:	84.81	%	Calculated assay recovery
Variance:	3.20		Variance of the assay recovery
SDs:	1.79	%	Standard deviation of the assay recovery

MassVar : Estimation of errors in two-product mass flowrate

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

MassVar calculates the error associated with the two-product recovery formula for the ***mass recovery***

Assay:		
Feed	45.14	%
Conc	50.29	%
Tail	28.72	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	0.95	%	0.43
Conc	0.93	%	0.47
Tail	0.93	%	0.27

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	4.64
B	3.53
C	
D	1.11

Yield:	76.12	%	Calculated mass recovery
Variance:	6.76		Variance of the recovery
SDs:	2.60	%	Standard deviation of the recovery

4.4.1.2.3. Analysis of the Excel Spreadsheets Results:

Equation 3.8 is useful in assessing the error that can be expected in the calculated value of recovery due to errors in the measurement of feed, concentrates, and tailings, The Magnetic Barrier Laboratory Separator, Model LB-1, has been used with current equals to 0.8 Amper and average flow rate for the thin fractions is equal to 20.32 g / minute to treat a feed containing 45.14 % Fe to produce a concentrate grading 50.29 % Fe and a tailing of 28.72 % Fe , the calculated value of recovery (Equation 3.6) is 84.81 , and by comparing an equation 3.8 , chapter 3, by an Excel Spreadsheets (Recovery Variance), we can obtain upon:

$$V_R = 0.01\,(1023.79V_f + 477.99V_c + 144.17\,V_t) \quad \text{.....(4.3)}$$

It is immediately apparent that the calculated value of recovery is most sensitive to the variance of the feed assay and the variance of the concentrates than the variance of the tailings assay. When, the feed, concentrate, and tailings have relative standard deviations of 0.95%, 0.93% and 0.93 %, respectively. Then the standard deviations of feed, concentrate, and

tailings are 0.43 %, 0.47 % and 0.27%, respectively, and, from Equation 4.3., V_R = 3.20, or the standard deviation of R is 1.79. This means that, to within 55.98 % confidence limits, the recovery is 84.81 ±1.79, and the concentrate is 50.29±0.47. Therefore, The value of C determined from iron ore assays (Equation 3.9, Chapter 3) is 76.12%, and, from (Equation 3.10, chapter 3), V c = 6.76. The standard deviation, s, is thus 2.6 %, and the relative standard deviation in the mass calculation *(s/C)* is 0.034, with 93.903 % confidence level.

also, we found that ,the concentrates of good quality can be obtained by grinding the ore within the size fraction of -125 to +75, as impurities such as MnO_2 is found to go into the tailings, as resulted of the washing process, which has been predated this process. Whereas, SO_4 contaminants have not reached the desirable level, but it is not problem at this stage, because, the main goal is to remove viable amounts of silica to the tailings stream.

4.4.1.3. Ferricrite and Bog1-001:

Table 4.29 shows the results of Ferricrete and Bog1-001 from the magnetic separation (Group1).

Size Fractions(n)	Feed		Concentrate		Tailings	
(µm)	Yield (%)	Assay, Fe %(S1.n)	Yield(%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-600 to+250	16.59	12.43	23.46	36.22	76.54	5.14
-250 to +125	42.54	11.07	20.17	45.68	79.83	2.33
-125 to +75	25.42	18.15	28.97	53.58	71.03	3.70
-75 to +53	15.44	17.39	22.49	53.79	77.51	6.83
Average		14.56	23.31	47.67	76.69	4.50

Table 4.30 shows the contaminants results of Ferricrete and Bog1-001 from magnetic separation(Group1).

Constituents,%	CS3.1	CS3.2	CS3.3	CS3.4
MnO_2	0.127	0.1304	0.2076	0.1798
CaO	0.083952	0.021128	0.032042	0.032042
SO4	0.12	0.07	0.17	0.09

Where:-

CS3.n Concentrates of sample (3), for size fraction (n).

TS3.n Tailings of sample (3), for size fraction (n).

S3.n Feed of sample (3), for size fraction (n).

Where, the range of size fractions are -600 to +250, -250 to +125, -125 to +75, and -75 to +53 µm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratio = F/C was calculated and found to be 4.29.

Where:-

F The yield of feed (%), C The yield of concentrate (%).

4.4.1.3.1. Analysis of the Statistical Parameters Results:

The average values of the assays of the feed, concentrates and tailings are 13.88 %, 49.9 % and 4.71 %, respectively, and the standard deviation of the total assays of the feed, concentrates and tailings are 0.68 %, 2.23 % and 0.21 %. With 95 % level of confidence we find, the value of t = 2.571. We can then calculate:

$(t * SDf)/ n^{1/2} = 0.71$ %, and

$(t * SDc)/ n^{1/2} = 2.34$ %, and

$(t * SDt)/ n^{1/2} = 0.22$ %

These results determine a range of 13.88 ± 0.71 %, 49.9 ± 2.34 % and 4.71 ± 0.22 %, for the feed, concentrates and tailings respectively, when we fixed a level of confidence of 95 %, in which we have a 95% chance of finding the true average. There is probably a systematic error in these experiments. The value of 49.9 + 2.34 % would be included in this interval and would thus be considered as a viable result.

4.4.1.3.2. An Excel Spreadsheets Results, (MassVar and RecVar):

RecVar : Estimation of errors in recovery calculations

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

RecVar calculates the error associated with the two-product recovery formula for the ***assay recovery***

Assay:		
Feed	13.88	%
Conc	49.9	%
Tail	4.71	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	4.9	%	0.68
Conc	4.46	%	2.23
Tail	4.46	%	0.21

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	0.03
B	286.72
C	0.91
D	1581.99

Recovery:	72.95	%	Calculated assay recovery
Variance:	5.26		Variance of the assay recovery
SDs:	2.29	%	Standard deviation of the assay recovery

MassVar : Estimation of errors in two-product mass flowrate

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

MassVar calculates the error associated with the two-product recovery formula for the ***mass recovery***

Assay:		
Feed	13.88	%
Conc	49.9	%
Tail	4.71	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	4.9	%	0.68
Conc	4.46	%	2.23
Tail	4.46	%	0.21

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	2.21
B	0.45
C	
D	1.76

Yield:	20.29	%	Calculated mass recovery
Variance:	3.40		Variance of the recovery
SDs:	1.84	%	Standard deviation of the recovery

4.4.1.3.3. Analysis of the Excel Spreadsheets Results:

Equation 3.8 is useful in assessing the error that can be expected in the calculated value of recovery due to errors in the measurement of feed, concentrates, and tailings . The Magnetic Barrier Laboratory Separator, Model LB-1, has been used with current equals to 0.8 Amper and average flow rate for the thin fractions is equal to 25.66 g / minute to treat a feed containing 13.88 % Fe to produce a concentrate grading 49.9 % Fe and a tailing of 4.71 % Fe , the calculated value of recovery (Equation 3.6) is 72.95 , and by comparing an equation 3.8, chapter 3, by an Excel Spreadsheets (Recovery Variance), we can obtain upon:

$$V_R = 0.03\,(286.72V_f + 0.91V_c + 1581.99V_t) \quad \text{.....(4.4)}$$

It is immediately apparent that the calculated value of recovery is most sensitive to the variance of the tailings assay and the variance of the feed than the variance of the concentrates assay. When, the feed, concentrates, and tailings have relative standard deviations of 4.9 %, 4.46 % and 4.46 %, respectively. Then the standard deviations of feed, concentrate, and tailings are 0.68, 2.23, and 0.21 % respectively, and, from Equation 4.4.,

V_R = 5.26, or the standard deviation of R is 2.29. This means that, to within 55.97984 % confidence limits, the recovery is 72.95 ±2.29, and the concentrate is 49.9±2.23 %.

Therefore, The value of C determined from iron ore assays (Equation 3.9, Chapter 3) is 20.29 %, and, from (Equation 3.10, chapter 3), V c = 3.40. The standard deviation, s, is thus 1.84 %, and the relative standard deviation in the mass calculation *(s/C)* is 0.091, with 93.903 % confidence level.

also, we found that ,the concentrates of good quality can be obtained by grinding the ore within the size fraction of -125 to + 75 um, Whereas, SO_4 contaminants have not reached the desirable level,but it is not problem at this stage, because, the main goal is to remove viable amounts of silica to the tailings stream.

4.4.1.4. Oolitic:

Table 4.31shows the results of Oolitic from the magnetic separation (Group1)

Size Fractions(n)	Feed		Concentrate		Tailings	
(µm)	yield (%)	Assay, Fe %(S1.n)	Yield(%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-600 to +250	11.20	52.08	73.54	55.00	26.46	43.98
-250 to +125	44.01	54.27	93.22	54.51	6.78	51.03
-125 to +75	28.04	46.89	82.60	47.52	17.40	43.94
-75 to +53	16.75	47.43	47.59	54.96	52.41	40.59
Average		50.79	80.39	52.76	19.61	42.71

Table 4.32 shows the contaminants results of Oolitic from magnetic separation (Group1)

Constituents,%	CS4.1	CS4.2	CS4.3	CS4.4
MnO_2	0.0761	0.0826	0.1183	0.1226
CaO	0.015251	0.01763	0.025046	0.024346
SO4	0.06	0.14	0.08	0.08

Where:-

CS4.n Concentrates of sample (4), for size fraction (n).

TS4.n Tailings of sample (4), for size fraction (n).

S4.n Feed of sample (4), for size fraction (n).

Where, the range of size fractions are -600 to +250, -250 to +125, -125 to +75, and -75 to +53 µm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratio = F/C was calculated and found to be 1.24.

Where:-

F The yield of feed (g), C The Yield of concentrate (g).

4.4.1.4.1. Analysis of the Statistical Parameters Results:

The average values of the assays of the feed, concentrates and tailings are 50.84 %, 52.81 % and 42.75 %, respectively, and the standard deviation of the total assays of the feed, concentrates and tailings are 0.05 %, 0.05 % and 0.04 %. With 95 % level of confidence we find, the value of t = 2.571. We can then calculate:

$(t * SDf)/ n^{1/2} = 0.05$ %, and

$(t * SDc)/ n^{1/2} = 0.05$ %, and

$(t * SDt)/ n^{1/2} = 0.04$ %

These results determine a range of 50.84 ± 0.05 %, 52.81 ± 0.05 % and 42.75 ± 0.04 %, for the feed, concentrates and tailings respectively, when we fixed a level of confidence of 95 %, in which we have a 95% chance of finding the true average. There is probably a systematic error in these experiments. The value of 52.81 + 0.05 % would be included in this interval and would thus be considered as a viable result.

4.4.1.4.2. An Excel Spreadsheets Results, (MassVar and RecVar):

RecVar : Estimation of errors in recovery calculations

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

RecVar calculates the error associated with the two-product recovery formula for the ***assay recovery***

Assay:		
Feed	50.84	%
Conc	52.81	%
Tail	42.75	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	0.09	%	0.05
Conc	0.1	%	0.05
Tail	0.1	%	0.04

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	0.04
B	1971.94
C	1181.88
D	106.95

Recovery:	83.53	%	Calculated assay recovery
Variance:	0.29		Variance of the assay recovery
SDs:	0.54	%	Standard deviation of the assay recovery

MassVar : Estimation of errors in two-product mass flowrate

B.A.Wills (1984)
Updated to MS Excel, JKTech Pty Ltd (2005)

MassVar calculates the error associated with the two-product recovery formula for the ***mass recovery***

Assay:		
Feed	50.84	%
Conc	52.81	%
Tail	42.75	%

Enter the Feed, Concentrate and Tail assays in the highlighted cells.

SDs:	**Relative %**		Absolute
Feed	0.09	%	0.05
Conc	0.1	%	0.05
Tail	0.1	%	0.04

Enter the relative error for the Feed, Concentrate and Tail assays in the highlighted cells.

Calculation	
A	9.94
B	7.99
C	
D	1.95

Yield:	80.42	%
Variance:	0.39	
SDs:	0.63	%

Calculated mass recovery
Variance of the recovery
Standard deviation of the recovery

4.4.1.4.3. Analysis of the Excel Spreadsheets Results:

Equation 3.8 is useful in assessing the error that can be expected in the calculated value of recovery due to errors in the measurement of feed, concentrates, and tailings. The Magnetic Barrier Laboratory Separator, Model LB-1, has been used with current equals to 0.8 Ampere and average flow rate for the thin fractions is equal to 12.37 g / minute, to treat a feed containing 50.84 % Fe to produce a concentrate grading 52.81 % Fe and a tailing of 42.75 % Fe , the calculated value of recovery (Equation 3.6) is 83.53 , and by comparing an equation 3.8, chapter 3, by an Excel Spreadsheets (Recovery Variance), we can obtain upon:

$$V_R = 0.04\ (1971.94\ V_f + 1181.88 V_c + 106.95 V_t) \quad \text{.....(4.5)}$$

It is immediately apparent that the calculated value of recovery is most sensitive to the variance of the feed assay and the variance of the concentrates than the variance of the tailings assay. When, the feed, concentrates, and tailings have relative standard deviations of 0.09 %, 0.1 % and 0.1 %, respectively. then the standard deviations of feed, concentrate, and tailings are 0.05, 0.05, and 0.04 % respectively, and, from Equation 4.5., $V_R = 0.29$, or

the standard deviation of R is 0.54. This means that, to within 55.98 % confidence limits, the recovery is 83.53 ±0.54, and the concentrate is 52.81 ±0.05. Therefore, The value of C determined from iron ore assays (Equation 3.9, Chapter 3) is 80.42 %, and, from (Equation 3.10, chapter 3), V c = 0.39. The standard deviation, s, is thus 0.63 %, and the relative standard deviation in the mass calculation *(s/C)* is 0.008, with 93.903 % confidence level.

also, we found that , feed with good quality can be obtained by grinding the ore within the size fractions of -250 to +125, as impurities such as MnO_2, and SO_4 were found to go into the tailings, as resulted of the washing process, which has been predated this process, also, we found that the total assay of the tailings are similar to the total assay of the concentrates, in this case, we can say, Oolitic iron samples need to washing process only.

4.4.2. Results of Group (2):

4.4.2.1. Bog1&Bog2:

Table 4.33 shows the results of Bog1 and Bog2 from the magnetic separation (Group2).

Size Fractions(n)	Feed		Concentrate		Tailings	
(µm)	Yield (%)	Assay, Fe %(S1.n)	Yield (%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-1180 t0+250	8.39	33.18	89.29	36.04	12.18	12.18
-250 to +125	34.71	38.27	87.21	39.22	2.30	2.30
-125 to +75	38.64	35.88	74.47	36.92	2.30	2.30
-75 to +53	18.26	35.86	79.40	37.97	5.53	5.53
Average		32.03	81.04	37.54	18.96	8.50

Where:-

CS1.n Concentrates of sample (1), for size fraction (n).

TS1.n Tailings of sample (1), for size fraction (n).

S1.n Feed of sample (1), for size fraction (n).

Where, the range of size fractions are -1,180 to +250, -250 to +125, -125 to +75, and -75 to +53 µm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratio = F/C was calculated and found to be 1.23.

Where:-

F The yield of feed(g), C The yield of concentrate(g).

Recovery % = (C * c / F * f)*100 = 94.98 %.

Where:-

c Analysis of concentrate(%), f Analysis of feed(%).

4.4.2. 2. Bog2-5:

Table 4.34 shows the results of Bog2-5 from the magnetic separation (Group2)

Size Fractions(n)	Feed		Concentrate		Tailings	
(μm)	Yield (%)	Assay, Fe %(S1.n)	Yield (%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-853 to +250	12.22	46.57	96.09	46.97	3.91	36.70
-250 to +125	36.57	48.98	98.69	49.34	1.31	21.62
-125 to +75	33.67	43.12	81.81	51.85	18.19	3.84
-75 to + 53	17.54	39.18	75.47	49.76	24.53	6.64
Average		44.71	88.62	49.37	11.38	8.44

Where:-

CS2.n Concentrates of sample (2), for size fraction (n).

TS2.n Tailings of sample (2), for size fraction (n).

S2.n Feed of sample (2), for size fraction (n).

Where, the range of size fractions are -853 to +250, -250 to +125, -125 to +75, and -75 to +53 μm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratio = F/C was calculated and found to be 1.13.

Where:-

F The yield of feed(g), C The yield of concentrate(g).

Recovery % = (C * c / F * f)*100 = 97.86 %.

Where:-

c Analysis of concentrate(%), f Analysis of feed(%).

4.4.2.3. Ferricrete and Bog1-001:

Table 4.35 shows the results of Ferricrete and Bog1-001 from the magnetic separation (Group2).

Size Fractions(n)	Feed		Concentrate		Tailings	
(μm)	Yield (%)	Assay, Fe %(S1.n)	Yield (%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-600 to +250	17.26	10.02	24.49	31.01	75.51	3.21
-250 to +125	43.36	9.17	23.16	36.99	76.84	0.79
-125 to +75	25.53	15.31	32.63	46.49	67.37	0.20
-75 to +53	13.86	20.73	41.82	46.79	58.18	2.00
Average		12.95	28.39	41.69	71.61	1.56

Where:-

CS3.n Concentrates of sample (3), for size fraction (n).

TS3.n Tailings of sample (3), for size fraction (n).

S3.n Feed of sample (3), for size fraction (n).

Where, the range of size fractions are -600 to +250, -250 to +125, -125 to +75, and -75 to +53 µm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratio = F/C was calculated and found to be 3.52.

Where:-

F The yield of feed (g), C The yield of concentrate (g).

Recovery % = (C * c / F * f)*100 = 91.40 %

Where:-

c Analysis of concentrate(%), f Analysis of feed(%).

4.4.2.4. Oolitic:

Table 4.36 shows the results of Oolitic from the magnetic separation (Group2).

Size Fractions(n)	Feed		Concentrate		Tailings	
(µm)	Yield (%)	Assay, Fe %(S1.n)	Yield (%), (CS1.n)	Assay, Fe %	Yield (%), (TS1.n)	Assay, Fe %
-600 to +250	11.39	48.62	87.69	49.38	12.31	43.24
-250 to +125	47.81	55.76	92.82	56.13	7.18	51.00
-125 to +75	27.21	46.15	98.02	46.70	1.98	19.10
-75 to +53	13.59	52.80	96.19	53.70	3.81	30.11
Average		50.88	94.11	51.46	5.89	41.57

Where:-

CS4.n Concentrates of sample (4), for size fraction (n).

TS4.n Tailings of sample (4), for size fraction (n).

S4.n Feed of sample (4), for size fraction (n).

Where, the range of size fraction are -600 to +250, -250 to +125, -125 to +75, and -75 to +53 µm, and the n values are represented by the number (1,2,3, and 4),respectively.

Concentration Ratio = F/C was calculated and found to be 1.06.

Where:-

F The Yield of feed (g), C The yield of concentrate(g).

Recovery % = (C * c / F * f)*100 = 95.18 %.

Where:-

c Analysis of concentrate (%), f Analysis of feed (%).

4.4.3. Conclusive Discussion of the Electromagnetic Separation Results for Group1 and Group2:

The advanced electromagnetic separator, which has several arranges of intensities, was used to separate iron oxide minerals from their gangue. When , the current of 0.8 ampere was applied into the samples of Group1, while the current of 1.0 ampere was applied into the samples of Group2, it is found out that, the concentration ratios, the yield of tailings and the concentrate grades are increased for all samples of Group1, but these were decreased for all samples of Group2,(Fig 4.1, 4.2 and 4.3), thus results were agreed with electromagnetic separators theories, The iron contents of tailings also are increased for the three samples of Group 1 and decreased for the three samples of Group2, but for the fourth sample, the iron content of tailings was increased in Group2 and decreased in Group1, (Fig 4.4), the interpretation for the errors in this sample refers to many reasons, first, For all chemical analyses, which refer to the species to be measured, they must be in a sufficient quantity and suitable form for the instrument used. The majority of samples require a specific pretreatment. This preliminary stage, which conditions instrument calibration and follows the so-called sampling procedure. It is often a critical step in a chemical analysis, because it has an influence upon the result. Secondly, the smaller portion for the sample of Group1 has not contained the same proportion of Group2 components and thirdly, the flow rate of the Group2 sample is higher than the flow rate of Group1 sample, (Fig 4.5). Also it is appeared that the recovery and the yield of concentrates have been increased for Group2 samples and decreased for Group1 samples, (Fig 4.6 and 4.7).

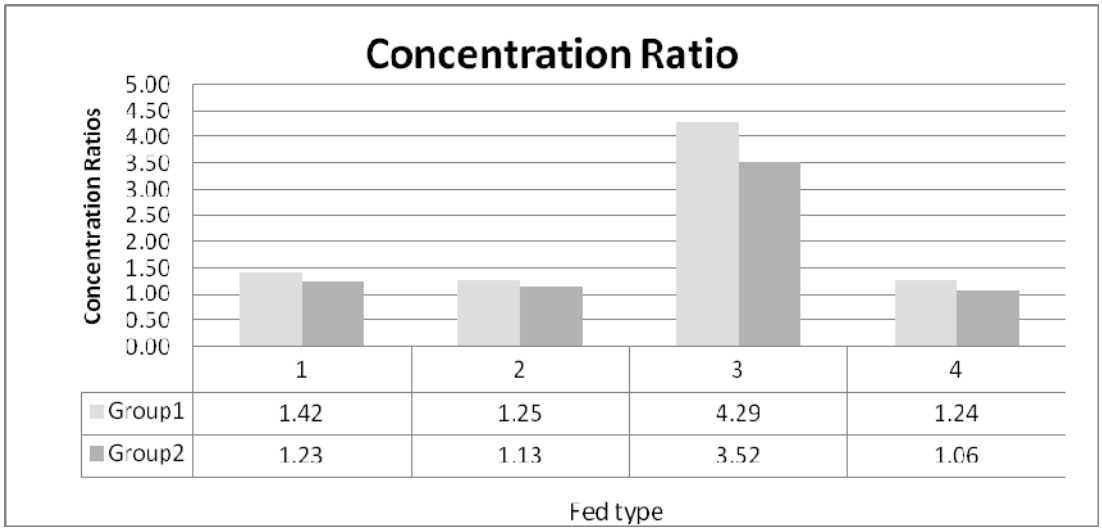


Fig. 4.1

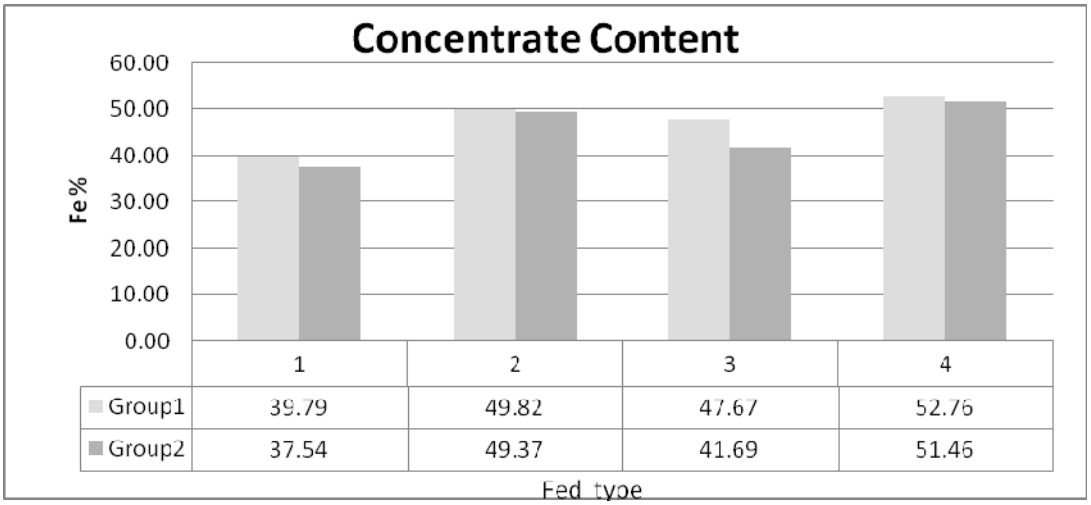


Fig. 4.2

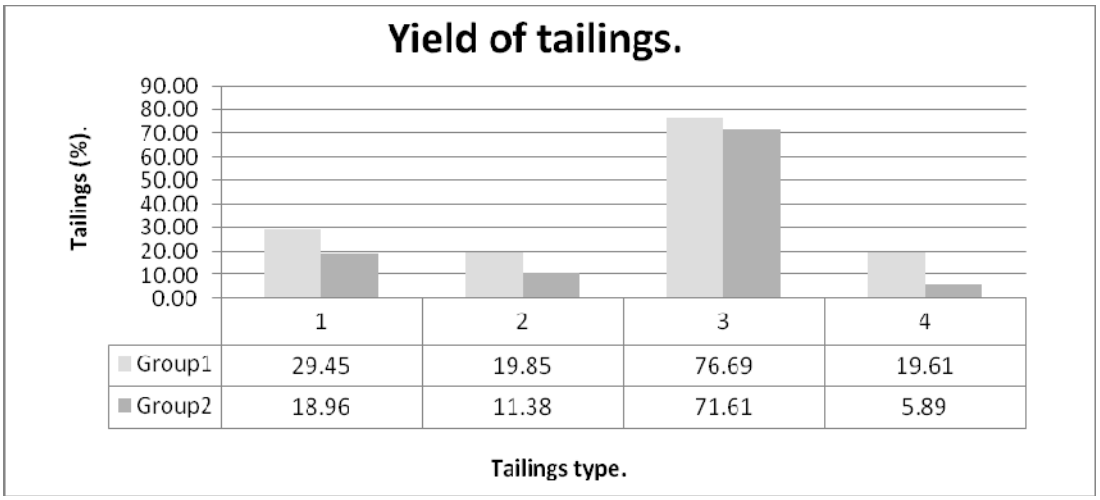


Fig. 4.3

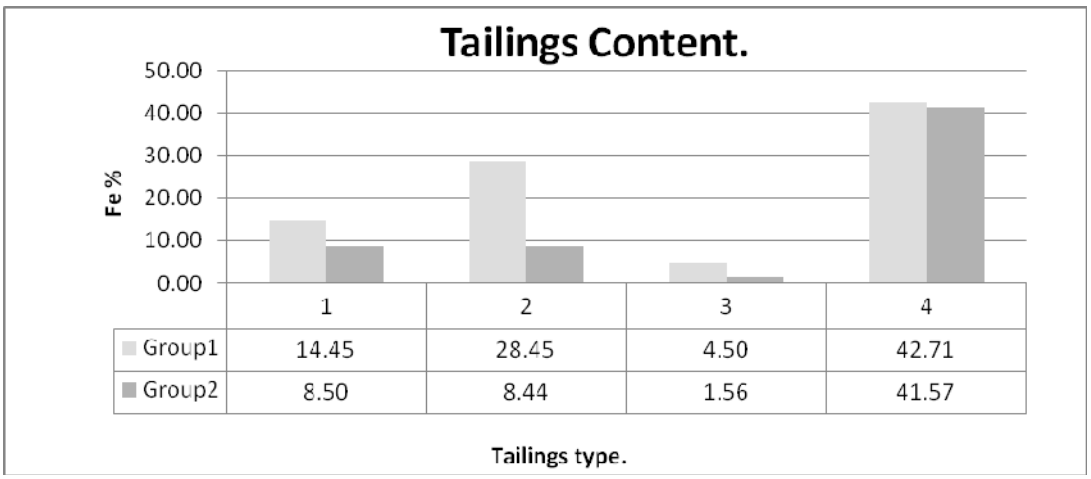


Fig. 4.4

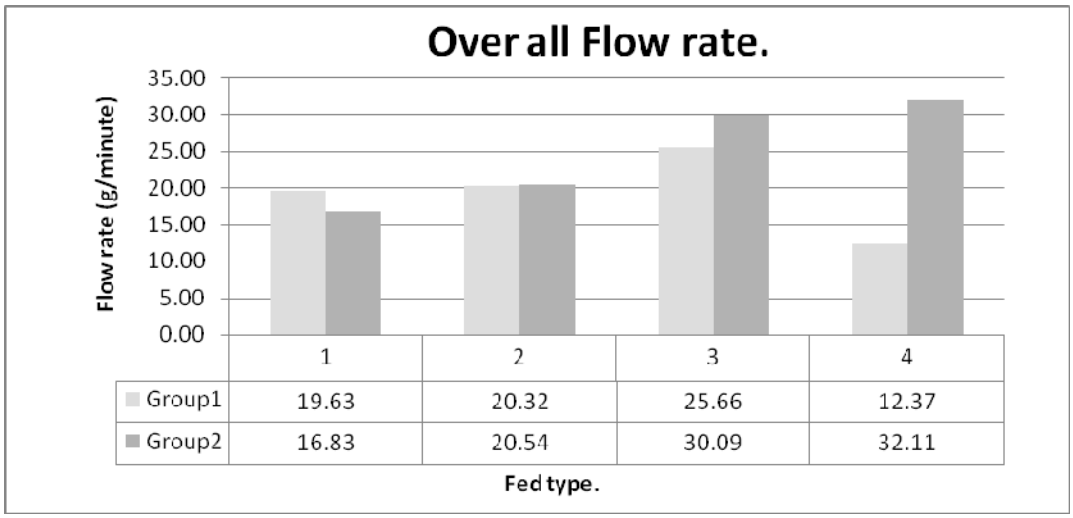


Fig.4.5

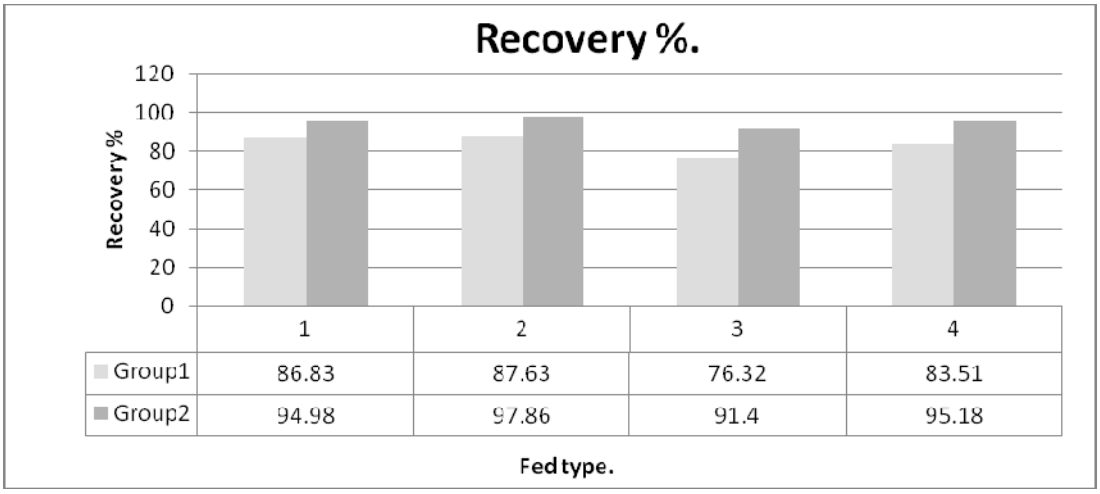


Fig. 4.6

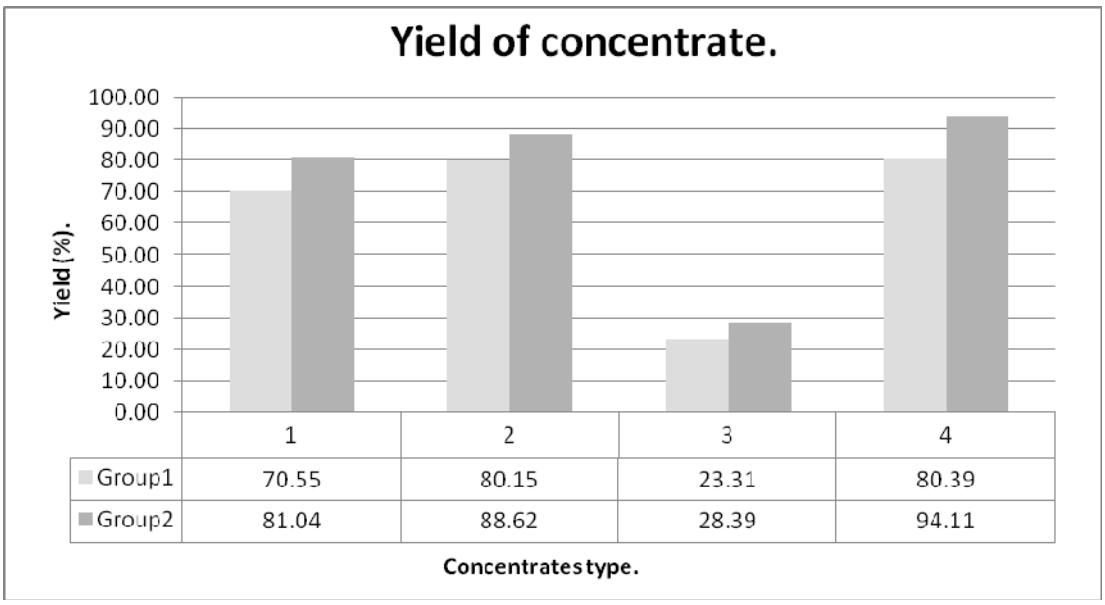


Fig. 4.7

4.4.4. Discussion of the Stereomicroscope Images (Group1):

From Plate 4.5 to 4.8, column 1 represents the washed samples for the different size fractions (Feed), column 2 represents the tailings, and column 3 represents the concentrates. It can be observed that there is not any significant difference in the color between all Oolitic ore fractions. At column 2, Plate 4.5, 4.6 and 4.7, it is observed, that much amounts of the free silica could be seen , within the size fractions of -250 to +53 micron, also, it is appeared that , there are more amounts of the iron within the size fraction of -75 to +53 μm, this means that the effectiveness of separation on such fine material is severely reduced by the effects of air currents, particle-particle adhesion, and particle-rotor adhesion, hence this results were agreed with the chemical analysis for these size fractions of ores, and with those opinions reported in the literatures, *(*Wills, Munn, 2006).

4.4.5. Discussion of the Results of the Solid Density for Group1 and Group2:

The tables below demonstrate the results of the solid density for group1 and Group2, which withdrawn from the concentrated iron ores samples.

Table 4.37shows the results of the concentrated solid density for Group1 and Group2.

Concentrates	Solid Density	
	Group1	Group2
Bog1&Bog2	3.55	3.52
Bog2-5	3.89	3.84
Ferricrete&Bog1 -001	3.65	3.63
Oolitic	3.91	3.91

Gravity analyses of the concentrates also have been made for each size fractions of Group1 and Group2. It is found out that when using a current of 0.8 and 1.0 ampere with Group1and Group2, respectively. The solid densities of the concentrate were higher for Group1 samples; however, they were lower for Group2 samples, (Fig. 4.8).

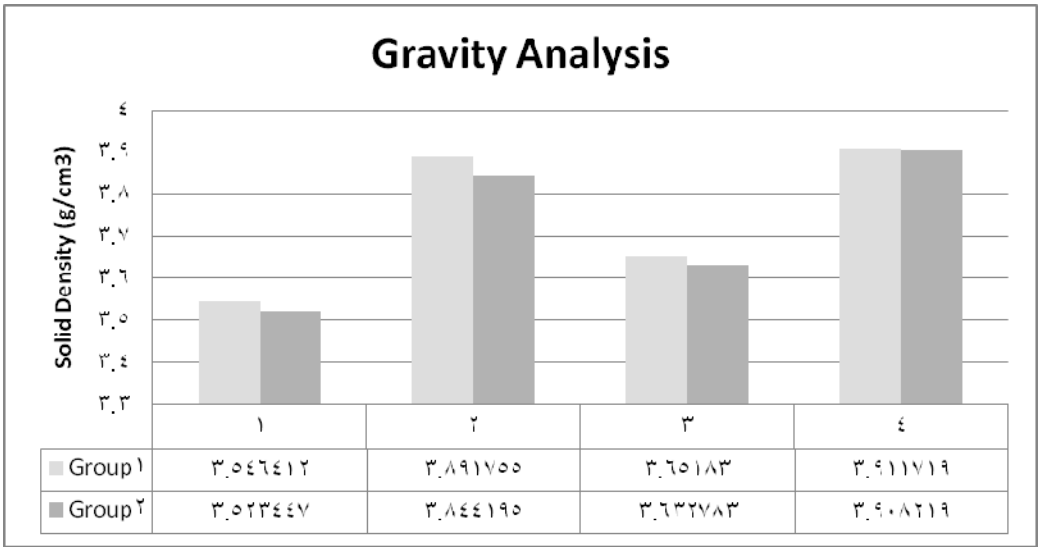
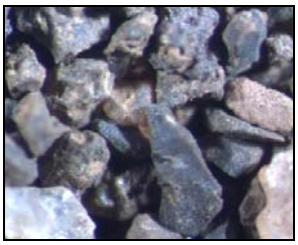


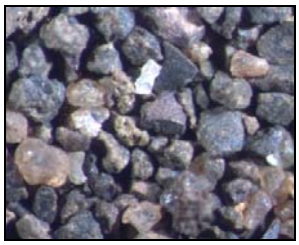
Fig. 4.8

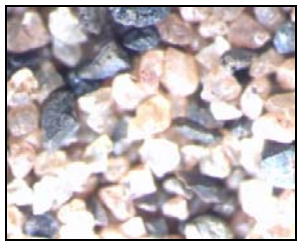
Plate 4.5- Images after washing and magnetic separation for the various size fractions of the samples.
1. Bog1and Bog2

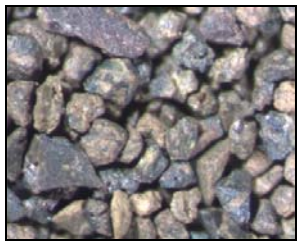

S1.1

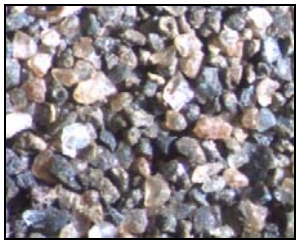

TS1.1

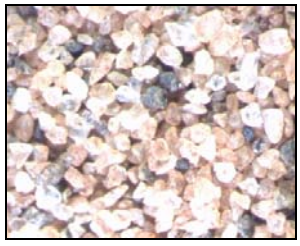

CS1.1


S1.2


TS1.2

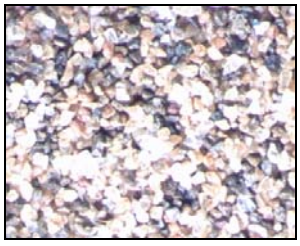

CS1.2


S1.3


TS1.3


CS1.3


S1.4


TS1.4


CS1.4

Plate 4.6- Images after washing and magnetic separation for the various size fractions of the samples.
2. Bog2-5

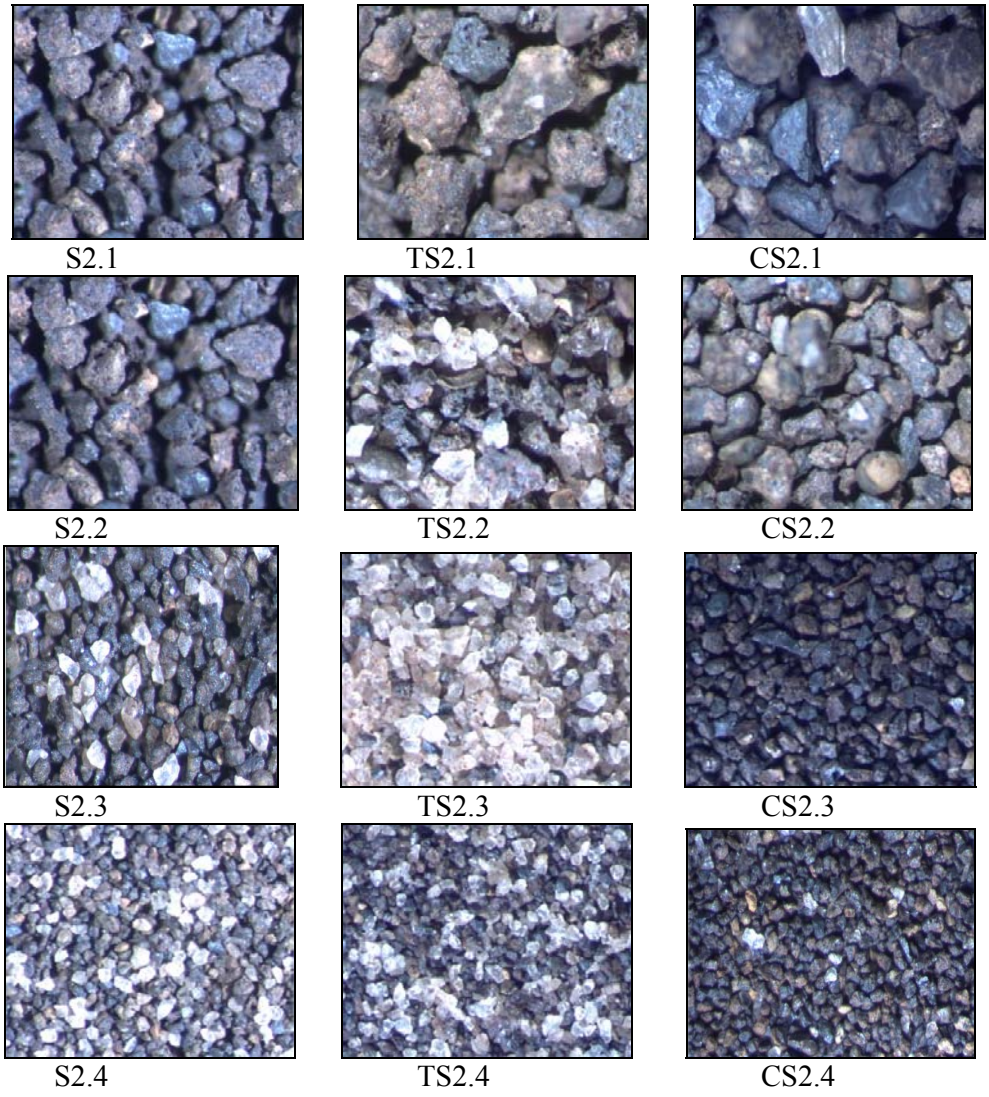


Plate 4.7- Images after washing and magnetic separation for the various size fractions of the samples.
3. Ferricrete and Bog1-001.

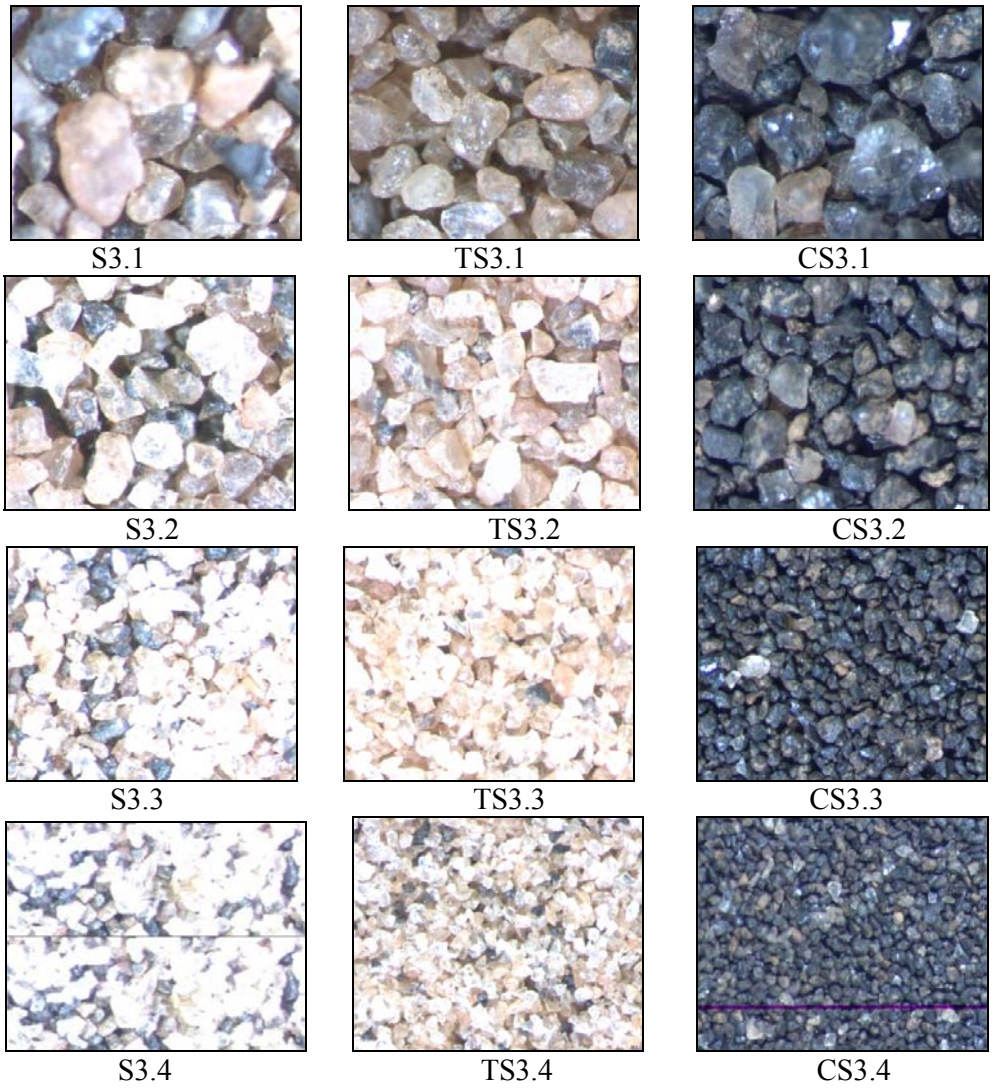
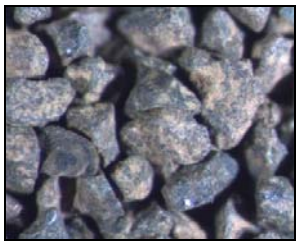
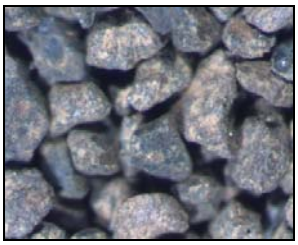
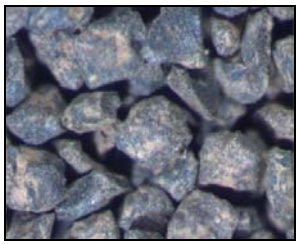


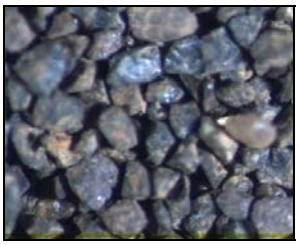
Plate 4.8- Images after washing and magnetic separation for the various size fractions of the samples.

4. Oolitic


S4.1

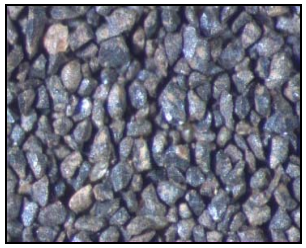

TS4.1


CS4.1


S4.2


TS4.2


CS4.2


S4.3


TS4.3


CS4.3


S4.4


TS4.4


CS4.4

4.6. Summary of the Results:

The results of the beneficiation of El Bajrawiya iron ores are summarized in Table 4.38 and Table 4.39. These results are also graphically shown in Fig 4.12 to Fig 4.19. The assessment of this work was performed on the basis of the grade of the concentrates and its yield and the recovery were gained. Three variables were considered to be of an utmost importance in upgrading of the iron ores of El Bajrawiya by dry high intensity magnetic process. They are: the size fraction of the feed, the intensity of the magnetic field which is assessed by the value of the ampere of the electric current producing the magnetic field, the feed rate for the magnetic separator. Therefore, a series of two test groups was conducted to determine the influence of the variables on the efficiency of the beneficiation process. The first group was carried out on four representative samples of the various ores where the strength of the electric current was 0.8 ampere. The average results of this group of tests are listed in table 4.40 and plotted in Fig. 4.12, Fig. 4.13, Fig. 4.14 and Fig. 4.15. The results show that:

Ferricrete and Bog1-001 mixture (1:1) which its feed assay was 14.56 % Fe gave concentrate yield % by weight 23.31 assaying 47.67 % Fe and the recovery of the iron was 76.32 %.

Bog1 and Bog2 mixture (1:1) its feed assaying 32.33 % Fe gave concentrate yield % around 70.55 % and assaying 39.79 % Fe.

The iron ore of Bog2-5 resulted concentrate its grade is 49.82 % Fe with a yield is 80.15 %, from feed its assay was 45.57 % Fe.

The Oolitic ore produced a yield concentrate of about 80.39 %, assaying 52.76 % Fe from feed assaying 50.79 % Fe.

Group2 of tests gave higher yields and recoveries, however, lower grades than the obtained results from group1 as a result of the increase in the value of the electric current from 0.8 ampere to 1.0 ampere.

For the same feeds grades, increasing the current to 1.0 ampere resulted concentrate yields 28.39 %, 81.04 %, 88.62 % and 94.11 %, having grades 41.69 %, 37.54 %, 49.37 %, 51.46 % Fe, and recoveries 91.40 %,94.98 %, 97.86 % and 95.18 %, respectively.

Concluding this summary of the results gained from the beneficiation of El Bajrawiya iron ores by a dry high intensity magnetic separator, grinding the ores to less than 250 μm is a must in order to liberate almost all iron oxide grains. Washing the ground ores to remove

the free fine silica and clay materials. The materials should be classified to remove the sizes lesser than 53 μm. The over sizes should be then dried and fed to the separator for upgrading.

Table 4.38 shows values of yield, assay and recovery for the concentrate and tailings, respectively, for four types from El Bajrawiya iron ores using dry electromagnetic separation process at 0.8 Ampere.
Group1.

Ore type	Concentrate			Tailings		
	Yield (%)	Assay (%)Fe	Recovery (%)	Yield (%)	Assay (%)Fe	Recovery (%)
Ferricrete&Bog1-001	23.31	47.67	76.32	76.69	4.5	23.68
Bog1&Bog2	70.55	39.79	86.83	29.45	14.45	13.17
Bog2-5	80.15	49.82	87.63	19.85	28.45	12.37
Oolitic	80.39	52.76	83.51	19.61	42.71	16.49

Table 4.39 shows values of yield, assay and recovery for the concentrate and tailings, respectively, for four types from El Bajrawiya iron ores using dry electromagnetic separation process at 1.0 Ampere.
Group2.

Ore type	Concentrate			Tailings		
	Yield (%)	Assay (%)Fe	Recovery (%)	Yield (%)	Assay (%)Fe	Recovery (%)
Ferricrete&Bog1-001	28.39	41.69	91.40	71.61	1.56	8.60
Bog1&Bog2	81.04	37.54	94.98	18.96	8.50	5.02
Bog2-5	88.62	49.37	97.86	11.38	8.44	2.14
Oolitic	94.11	51.46	95.18	5.89	41.57	4.82

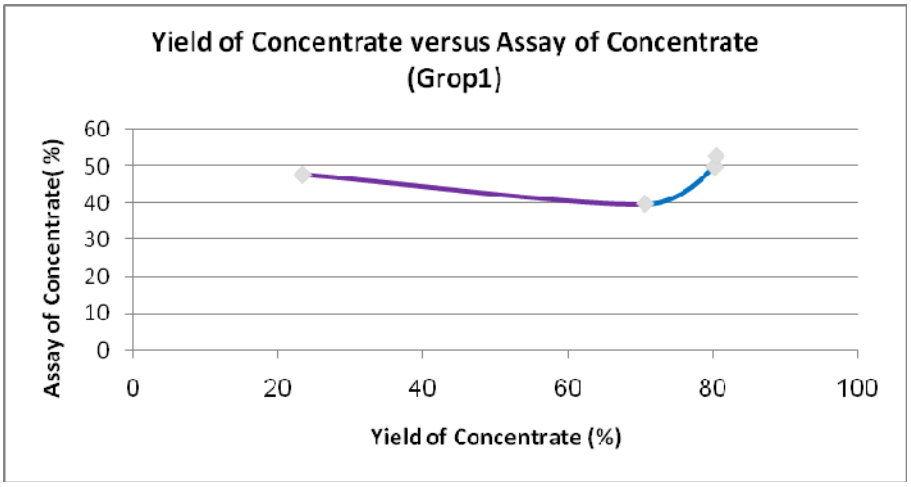


Fig. 4.12

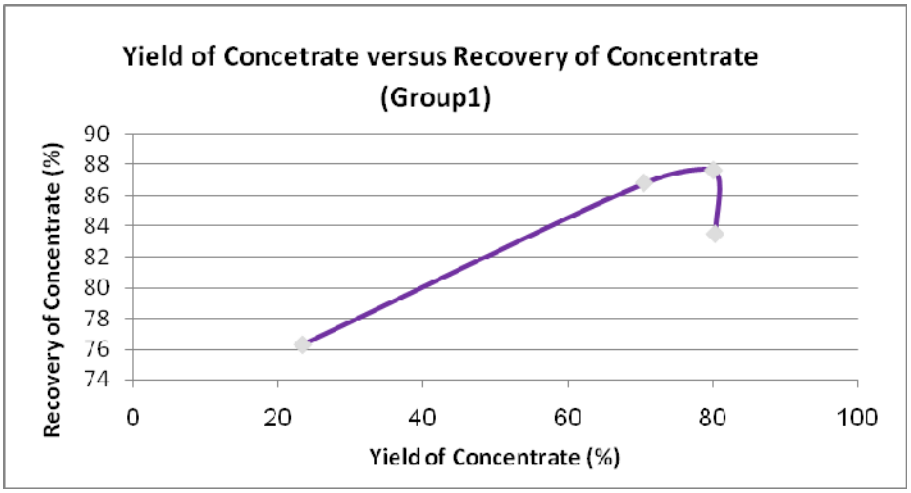


Fig. 4.13

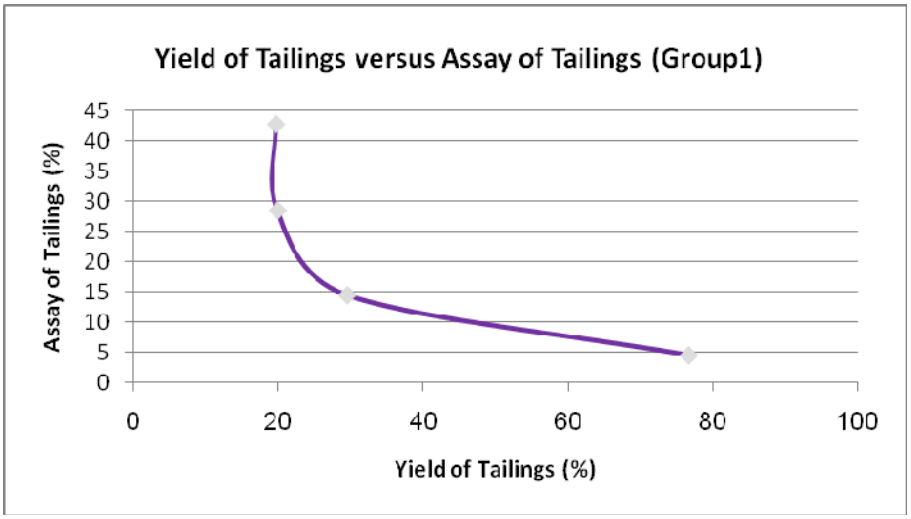


Fig. 4.14

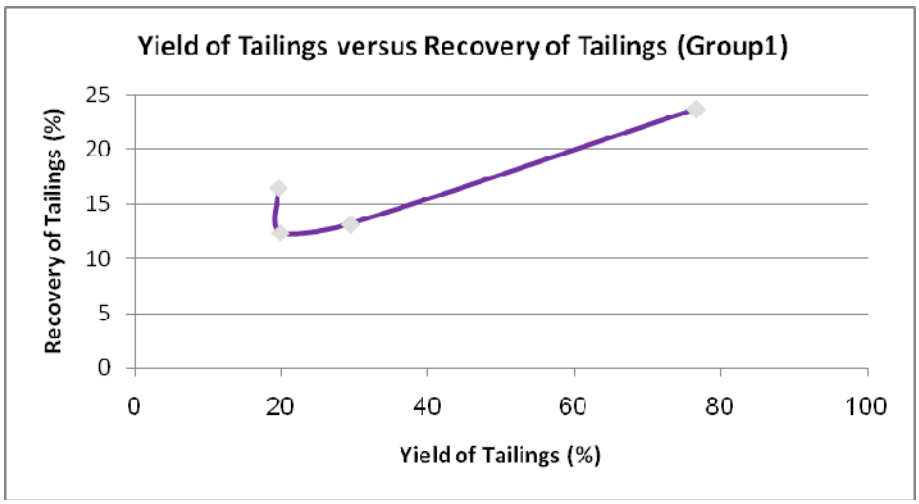


Fig. 4.15

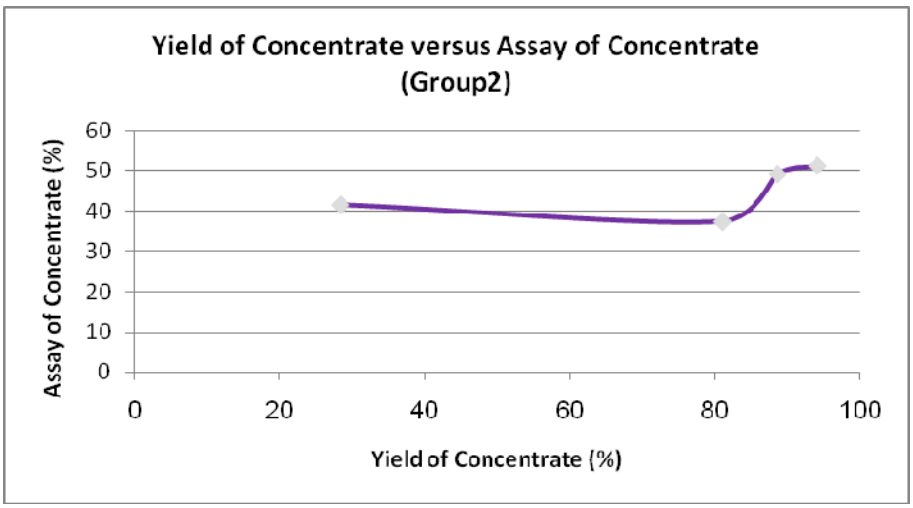


Fig. 4.16

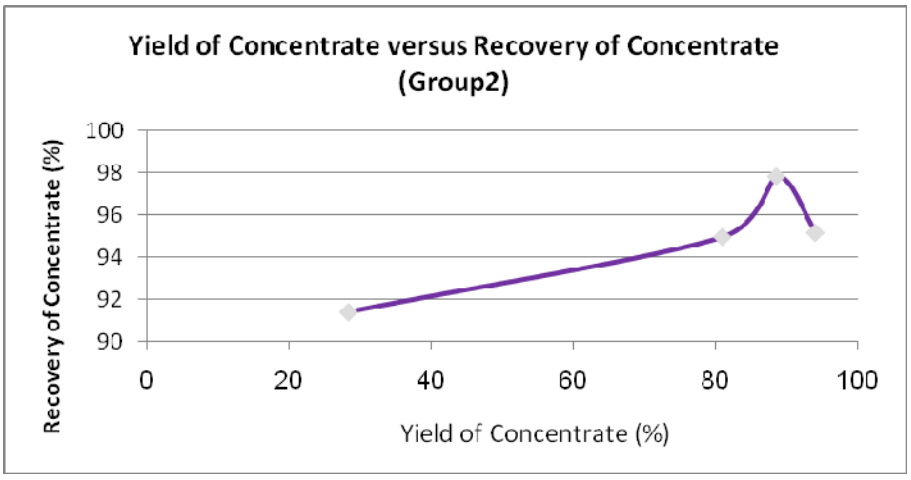


Fig. 4.17

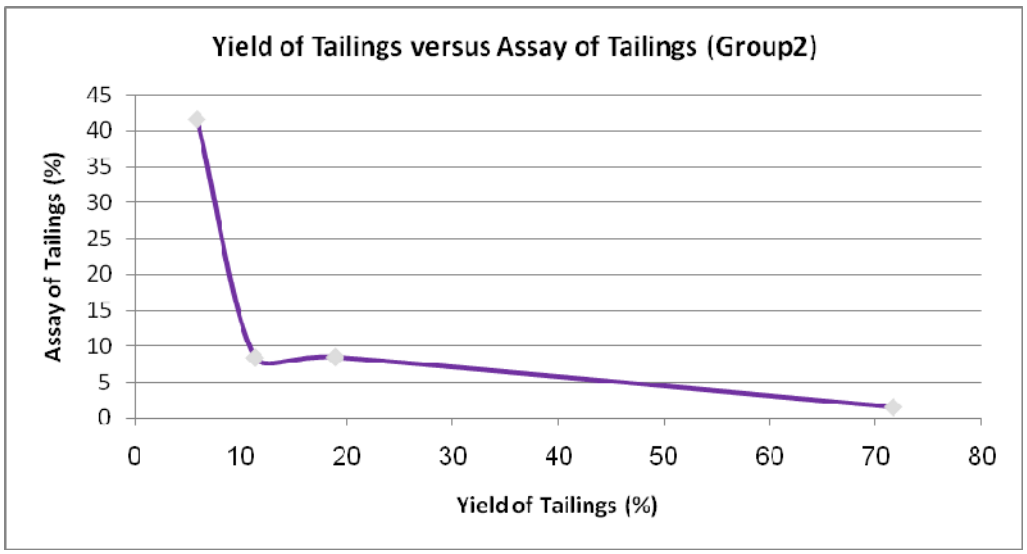


Fig. 4.18

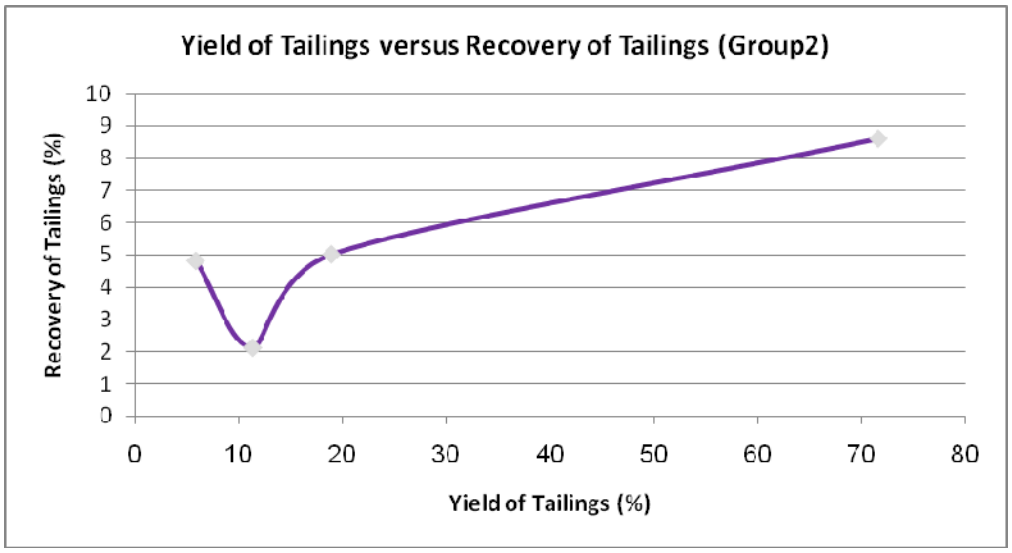


Fig. 4.19

Chapter Five

Conclusions and Recommendations

5.1. Conclusions

Beneficiation of El Bajrawiya iron ores in the North State of Sudan was investigated using dry high intensity magnetic separation process. Six types of iron oxides were processed. They are named as Bog1, Bog2, Bog2-5, Ferricrete, Bog1-001 and Oolitic ore. Samples of the outcrops of the deposits for these ores were taken by trenching method. From these outcrops it could be seen that there are a very thin layers separating Bog1and Bog2, and the Ferricrete and Bog1-001, therefore, mixtures of Bog1 with Bog2 and Ferricrete with Bog1-001, were made to be fed to the magnetic separation machine. Before subjecting each material to the magnetic separation, it was prepared for the separation process, by crushing, grinding, and washing to remove the ultrafine particles which are not suitable for dry magnetic processing. The screen used was a 53 μm screen. The size distribution analysis of these ground materials has indicated that the ores of Bog1 with Bog2 and Bog2-5 are softer than of the Ferricrete with Bog1-001 and the Oolitic, since the amounts of fine particles (-53 μm) were greater for Bog1 with Bog2 and Bog2-5 than those for the Ferricrete with Bog1-001 and the Oolitic ores. The softer ores have resulted, 40 % and 37 % coarser yields (+53 μm), while the harder ores resulted 51 % and 53 % yields, respectively. The coarser materials were processed by the dry high intensity magnetic method. The following conclusions could be withdrawn from the obtained results.

5.1.1. All types of ores with the exception of Oolitic ore could be upgraded by the dry high intensity magnetic separation.

5.1.2. The iron content were increased:

a) For Bog1 and Bog2 mixture from 32.33 % (washed feed) to 39.79 % in the concentrate.

b) For Bog2-5 ore from 45.57 % (washed feed) to 49.82 % for the concentrated material.

c) For Ferricrete and Bog1-001 from 14.56 % (washed feed) to 47.67 in the concentrate.

d) For Oolitic ore from 50.79% (washed feed) to 52.76 % of concentrate.

5.1.3. The recoveries of these various ores were about 86.83, 87.63, 76.32 and 83.51 % respectively for magnetic field 0.8 ampere.

5.1.4. when the ampere of the magnetic field was increased from 0.8 to 1.0 ampere, the iron content of concentrates produced from all ores were lesser than of those for 0.8 ampere, however, the recoveries were higher than that of the previous recoveries.

The iron contents of the concentrates for 1.0 ampere were 37.54 %, 49.37 %, 41.69 % and 51.46 %, however, the recoveries were about 94.98%, 97.86 %, 91.40 % and 95.18 % respectively.

5.2. Recommendations

For the future research work:

5.2.1. To upgrade the considerable amounts of fine resulted from the grinding process of these ores wet high intensity magnetic separation process is candidate for the (-53 μm) sizes.

5.2.2. A combination between dry high intensity magnetic process (for coarser particles) and froth floatation process could be used to beneficiate El Bajrawiya iron ores.

5.2.3. Grinding the ores of this area to sizes less than 70 μm and coarser than 10 μm in order to separate the iron oxides from the gangue materials by only froth floatation process.

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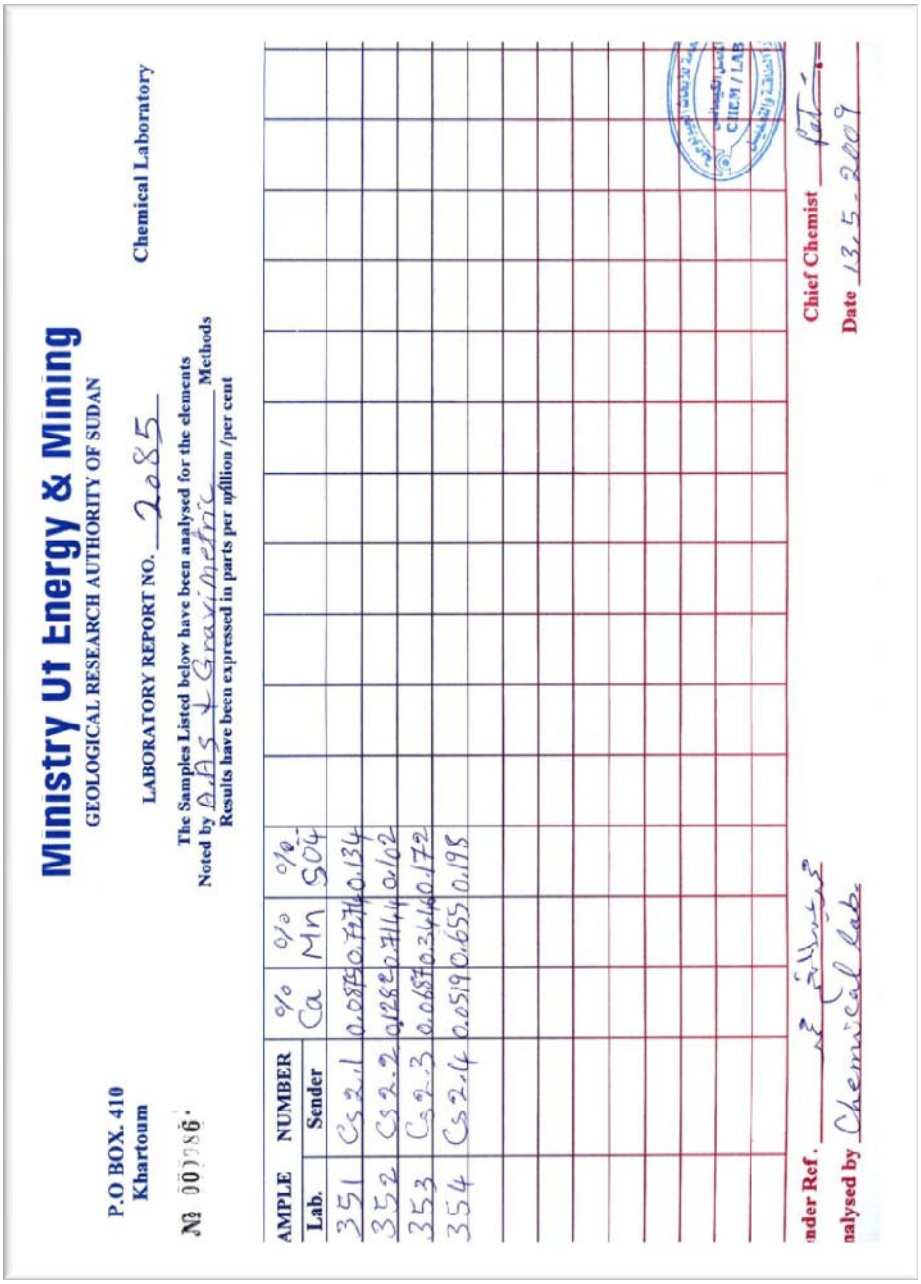
Appendix A The Chemicals Analyses of the Feed and Products.



Ministry Of Energy & Mining

GEOLOGICAL RESEARCH AUTHORITY OF SUDAN

P.O BOX. 410
Khartoum

Chemical Laboratory

LABORATORY REPORT NO. 2085

№ 00986

The Samples Listed below have been analysed for the elements
Noted by AAS & Gravimetric Methods
Results have been expressed in parts per million /per cent

AMPLE NUMBER		%	%	%
Lab.	Sender	Ca	Mn	SO4
351	Cs 2.1	0.0859	0.7374	0.134
352	Cs 2.2	0.1266	0.7144	0.102
353	Cs 2.3	0.0687	0.3546	0.172
354	Cs 2.4	0.0519	0.655	0.198

CHEM / LAB

Chief Chemist

Date 13-5-2009

Under Ref. _____

Analysed by Chemical Lab.

Ministry Of Energy & Mining

GEOLOGICAL RESEARCH AUTHORITY OF SUDAN

P.O BOX. 410
Khartoum

№ 00285

LABORATORY REPORT NO. 2085

Chemical Laboratory

The Samples Listed below have been analysed for the elements
Noted by AAS & Gravimetric **Methods**
Results have been expressed in parts per million /per cent

AMPLE NUMBER		%	%	%											
Lab.	Sender	Ca	Mn	$SO_4^=$	〃	〃	〃	〃	〃	〃	〃	〃	〃	〃	〃
253	Cs 1.1	0.0086	0.0804	0.13											
254	Cs 1.2	0.008	0.0724	0.14											
255	Cs 1.3	0.0065	0.0885	0.11											
256	Cs 1.4	0.0048	0.0664	0.05											
261	CS 4.1	0.0109	0.0761	0.06											
262	Cs 4 2	0.0126	0.0826	0.14											
263	CS 4.3	0.0179	0.1183	0.08											
264	CS 4.4	0.0174	0.1226	0.08											
243	Cs 3.1	0.06	0.127	0.12											
244	Cs 3.2	0.0151	0.1304	0.07											
245	Cs 3.3	0.0229	0.2076	0.17											
246	CS 3.4	0.0229	0.1798	0.09											
〃	〃	〃	〃	〃											

CHEM / LAB

Sender Ref. محمد عبدالمنعم محمد

Chief Chemist

Analysed by Chemical Lab.

Date 13.5.2006

BOX 410
rtoum

LABORATORY REPORT NO. 2103

Chemical Lat

The Samples Listed below have been analysed for the elements
Noted by A.A.S & Gravimetric **Methods**

Results have been expressed in parts per million / per cent

AMPLE NUMBER		%	%	%	%	〃	〃	〃	〃	〃
Lab.	Sender	Fe	Mn	Ca	SO_4	〃	〃	〃	〃	〃
519	Bog 1	34.03	0.081	0.249	0.731	〃				
520	Bog 2	31.57	0.064	0.383	0.600	〃				
521	Bog-001	20.76	0.115	0.228	0.096	〃				
522	Bog_2-5	41.13	1.47	0.591	0.320	〃				
523	Oolitic	48.00	0.14	0.221	0.207	〃				
524	S_1-5	〃	0.081	0.25	0.113	〃				
525	S_2-5	〃	0.965	0.50	0.247	〃				
526	S_3-5	〃	0.14	0.25	0.090	〃				
527	S_4-5	〃	0.146	0.195	0.279	〃				
528	Fernicrete	13.24	0.158	0.175	0.358	〃				
〃	〃	〃	〃	〃	〃	〃				

العمل الكيميائي
CHEM / LAB

Sender Ref. محمد مصطفى محمد

Chief Chemist. Sathi

Analysed by Chemical Lab

Date 7. 12. 2009

BOX 410
rtoum

Chemical Labo

LABORATORY REPORT NO. 2085 (Group 2)

The Samples Listed below have been analysed for the elements

Noted by AAS **Methods**

Results have been expressed in parts per million / per cent

AMPLE NUMBER		%	〃	〃	〃	〃	〃	〃	〃	〃	〃
Lab.	Sender	Fe	〃	〃	〃	〃	〃	〃	〃	〃	
359	Cs_2-1	46.97	〃								
360	Cs_2-2	49.34	〃								
361	Cs_2-3	51.85	〃								
362	Cs_2-4	49.76	〃								
363	$\overline{T}s_2-1$	36.70	〃								
364	$\overline{T}s_2-2$	21.62	〃								
365	$\overline{T}s_2-3$	3.84	〃								
366	$\overline{T}s_2-4$	8635	〃								
367	Cs_3-1	31.01	〃								
368	Cs_3-2	36.99	〃								
369	Cs_3-3	46.49	〃								
370	Cs_3-4	46.79	〃								
371	$\overline{T}s_3-1$	3.21	〃								
372	$\overline{T}s_3-2$	0.79	〃								
373	$\overline{T}s_3-3$	0.20	〃								
374	$\overline{T}s_3-4$	2.00	〃								
〃	〃	〃	〃								
〃	〃	〃	〃								

المختبر المركزي للأبحاث الجيولوجية
المعمل الكيميائي
CHEM. LAB
وزارة الطاقة والتعدين

Sender Ref. محمد المنجم

Chief Chemist.

Analysed by Chemical Lab

Date 6.4.2009

artoum

LABORATORY REPORT NO. 2076 (group2)

Chemical Lab

The Samples Listed below have been analysed for the elements

Noted by A.A.S **Methods**

Results have been expressed in parts per million / per cent

AMPLE NUMBER		%								
Lab.	Sender	Fe								
37	Cs1.1	36.04								
38	Cs1.2	39.22								
39	Cs1.3	36.92								
240	Cs1.4	37.97								
241	Ts1.1	12.18								
242	Ts1.2	2.296								
243	Ts1.3	2.296								
244	Ts1.4	5.534								
245	Cs4.1	49.38								
246	Cs4.2	56.13								
247	Cs4.3	46.69								
248	Cs4.4	53.7								
249	Ts4.1	43.24								
250	Ts4.2	51.00								
251	Ts4.3	19.10								
252	Ts4.4	30.11								
25	11	11								

الهيئة العامة للابحاث الجيولوجية
المعمل الكيميائي
CHEM / LAB

Sender Ref. عينات تربة

Chief Chemist

Analysed by. Chemical

Date 30.3.2009

BOX 410
ırtoum

LABORATORY REPORT NO. 2087 (group1)

Chemical Lat

The Samples Listed below have been analysed for the elements

Noted by A.A.S **Methods**

Results have been expressed in parts per million / per cent

AMPLE NUMBER		%	〃	〃	〃	〃	〃	〃	〃	〃
Lab.	Sender	Fe	〃	〃	〃	〃	〃	〃	〃	〃
343	Cs3-1	36.22	〃							
344	Cs3-2	45.68	〃							
345	Cs3-3	53.58	〃							
346	Cs3-4	53.79	〃							
347	Ts3-1	5.14	〃							
348	Ts3-2	2.33	〃							
349	Ts3-3	3.70	〃							
350	Ts3-4	6.83	〃							
351	Cs2-1	46.2	〃							
352	Cs2-2	49.5	〃							
353	Cs2-3	53.53	〃							
354	Cs2-4	50.97	〃							
355	Ts-1	42.24	〃							
356	Ts-2	28.08	〃							
357	Ts-3	12.59	〃							
358	Ts-4	30.87	〃							
〃	〃	〃	〃							
〃	〃	〃	〃							

CHEM / LAB

Sender Ref. [illegible]

Chief Chemist

Analysed by Chemical Lab

Date 7.5.2019

BOX 410
ırtoum

LABORATORY REPORT NO. 2078 (group 1) **Chemical La**

The Samples Listed below have been analysed for the elements
Noted by A.A.S **Method:**

Results have been expressed in parts per million / per cent

AMPLE NUMBER		010	//	//	//	//	//	//	//	//
Lab.	Sender	Fe	//	//	//	//	//	//	//	//
253	Cs 1.1	37.97								
254	Cs 1.2	41.34								
255	Cs 1.3	37.85								
256	Cs 1.4	42.51								
257	TS 1.1	10.24								
258	TS 1.2	15.56								
259	TS 1.3	10.89								
260	TS 1.4	18.60								
261	Cs 4.1	55.00								
262	Cs 4.2	54.51								
263	Cs 4.3	47.515								
264	Cs 4.4	54.96								
265	TS 4.1	43.98								
266	TS 4.2	51.03								
267	TS 4.3	43.94								
268	TS 4.4	40.59								
//	//	//								

المعمل الكيميائى CHEM / LAB

Sender Ref. عينات تشغ ٣

Chief Chemist.

Analysed by Chemical D.L **Date** 28 2 2009

BOX 410
artoum

LABORATORY REPORT NO. 2076

Chemical Lal

The Samples Listed below have been analysed for the elements

Noted by A.A.S **Methods**

Results have been expressed in parts per million / per cent

AMPLE NUMBER		o/o	//	//	//	//	//	//	//	//
Lab.	Sender	Fe	//	//	//	//	//	//	//	//
269	S1.5	33.08								
270	S2.5	42.65								
271	S3.5	21.65								
272	S4.5	45.44								
//	//	//								

CHEM / LAB

Sender Ref. جامعة العلوم والتقانة **Chief Chemist.**

Analysed by. Chemical lab. **Date** 3.3.2009

Appendix B. The Chemical Elements: Standard Atomic Masses. Swaddle, (1997).

Element	*Symbol*	*Atomic Number*	*Atomic Mass*
Bromine	Br	35	79.904(1)
Cadmium	Cd	48	112.411(8)
Calcium	Ca	20	40.078(4)
Californium	Cf	98	[251]
Carbon	C	6	12.011(1)
Cerium	Ce	58	140.115(4)
Cesium	Cs	55	132.90543(5)
Chlorine	Cl	17	35.4527(9)
Chromium	Cr	24	51.9961(6)
Cobalt	Co	27	58.93320(1)
(Columbium)	(Cb)	see Niobium	
Copper	Cu	29	63.546(3)
(Cuprum)	Cu	see Copper	
Curium	Cm	96	[247]
Dysprosium	Dy	66	162.50(3)
Einsteinium	Es	99	[254]
Erbium	Er	68	167.26(3)
Europium	Eu	63	151.965(9)
Fermium	Fm	100	[253]
(Ferrum)	Fe	see Iron	
Fluorine	F	9	18.9984032(9)
Francium	Fr	87	[212]
Gadolinium	Gd	64	157.25(3)
Gallium	Ga	31	69.723(1)
Germanium	Ge	32	72.61(2)
Gold	Au	79	196.96654(3)
Hafnium	Hf	72	178.49(2)
Helium	He	2	4.002602(2)
Holmium	Ho	67	164.93032(3)
(Hydrargyrum)	Hg	see Mercury	
Hydrogen	H	1	1.00794(7)
Indium	In	49	114.818(3)
Iodine	I	53	126.90447(3)
Iridium	Ir	77	192.22(3)
Iron	Fe	26	55.847(3)
(Kalium)	K	see Potassium	
Krypton	Kr	36	83.80(1)
Lanthanum	La	57	138.9055(2)
Lawrencium	Lr	103	[262]
Lead	Pb	82	207.2(1)
Lithium	Li	3	6.941(2)
Lutetium	Lu	71	174.967(1)

Element	*Symbol*	*Atomic Number*	*Atomic Mass*
Magnesium	Mg	12	24.3050(6)
Manganese	Mn	25	54.93805(1)
Mendelevium	Md	101	[258]
Mercury	Hg	80	200.59(2)
Molybdenum	Mo	42	95.94(1)
(Natrium)	Na	see Sodium	
Neodymium	Nd	60	144.24(3)
Neon	Ne	10	20.1797(6)
Neptunium	Np	93	[237]
Nickel	Ni	28	58.6934(2)
Niobium	Nb	41	92.90638(2)
Nitrogen	N	7	14.00674(7)
Nobelium	No	102	[259]
Osmium	Os	76	190.23(3)
Oxygen	O	8	15.9994(3)
Palladium	Pd	46	106.42(1)
Phosphorus	P	15	30.973762(4)
Platinum	Pt	78	195.08(3)
(Plumbum)	Pb	see Lead	
Plutonium	Pu	94	[244]
Polonium	Po	84	[209]
Potassium	K	19	39.0983(1)
Praseodymium	Pr	59	140.90765(3)
Promethium	Pm	61	[145]
Protactinium*	Pa	91	231.03588(2)
Radium	Ra	88	[226]
Radon	Rn	86	[222]
Rhenium	Re	75	186.207(1)
Rhodium	Rh	45	102.90550(3)
Rubidium	Rb	37	85.4678(3)
Ruthenium	Ru	44	101.07(2)
Samarium	Sm	62	150.36(3)
Scandium	Sc	21	44.955910(9)
Selenium	Se	34	78.96(3)
Silicon	Si	14	28.0855(3)
Silver	Ag	47	107.8682(2)
Sodium	Na	11	22.989768(6)
(Stannum)	Sn	see Tin	
(Stibium)	Sb	see Antimony	
Strontium	Sr	38	87.62(1)
Sulfur	S	16	32.066(6)
Tantalum	Ta	73	180.9479(1)